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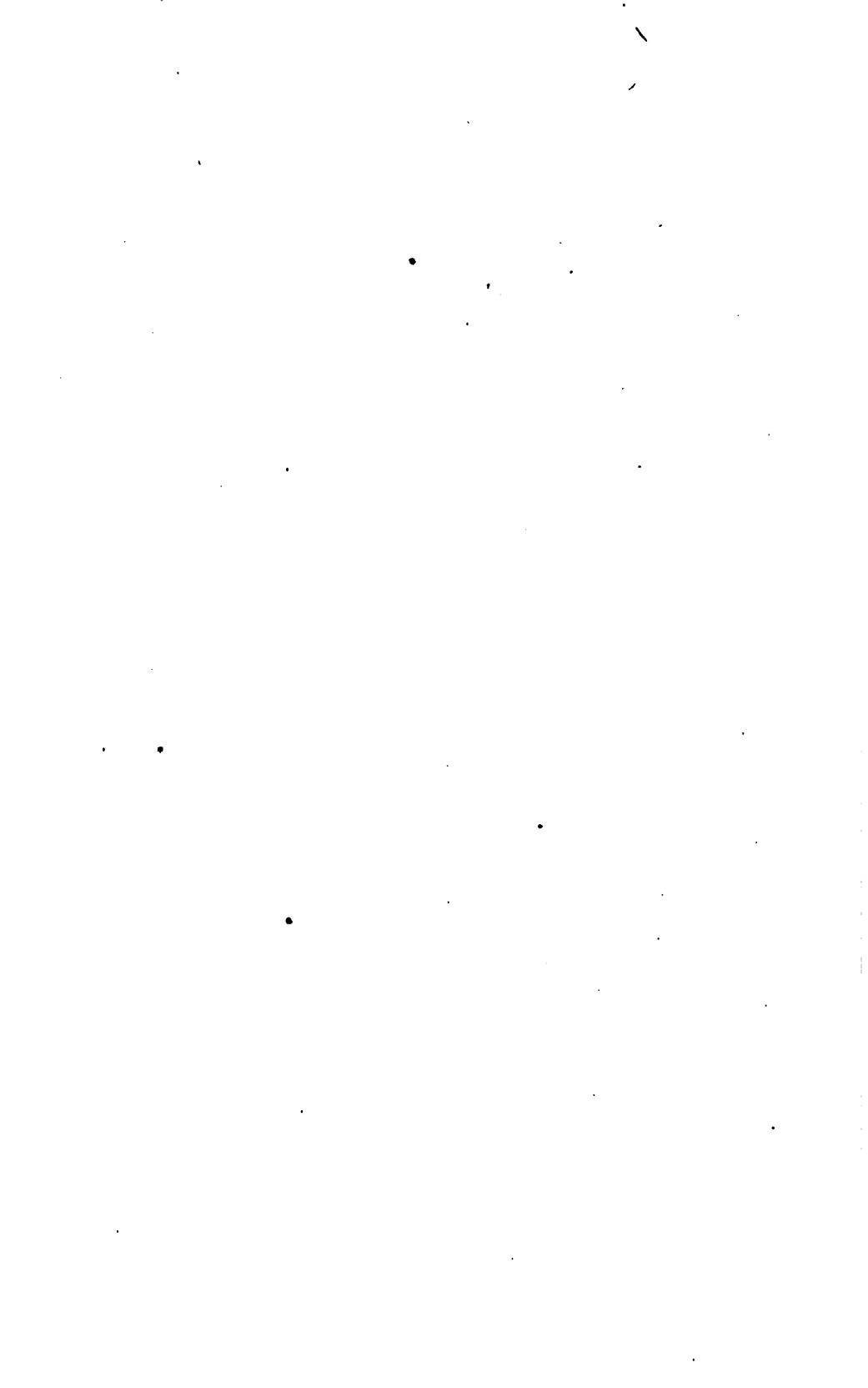
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JANUARY, 1915



Bulletin of the American Institute of Mining Engineers

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29 West 39th Street, New York, N. Y.

Mr. _____
(Name in Full)

Occupation _____

Address _____

is hereby proposed by the undersigned, as a _____

of the American Institute of Mining Engineers.

Signatures of three
Members or
Associates.

*Place of birth*_____ *Year of birth*_____

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

[illegible]

Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)

[illegible]

Present position

Signature.

Dated _____ 191

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 97

JANUARY

1915

Published Monthly by the American Institute of Mining Engineers at 312-318 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 30 West 29th St., New York, N. Y., BRADLEY SROUGHTON, Editor, F. O. PINNOC, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscriptions (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

SAMUEL FRANKLIN EMMONS MEMORIAL FELLOWSHIP

The friends of the late Dr. Samuel Franklin Emmons have established a fund whose income may be used in support of a fellowship to promote investigations in the branches of geology which were cultivated by him, more especially on the economic side. The funds have been placed in charge of the Trustees of Columbia University, but the choice of the fellow and the expenditure of the income are entrusted to a committee consisting of Professors James F. Kemp, John D. Irving, and Waldemar Lindgren. The committee announces that it will be prepared to award in March, 1915, a fellowship of \$1,000 for the year July 1, 1915, to June 30, 1916, inclusive. Applications must be made on blanks which will be furnished by the Secretary of Columbia University, New York, N. Y., and which when filled and accompanied by testimonials and complete statements of the applicant's qualifications will be submitted by him to the committee on March 1, 1915. Applications must be received by the Secretary of Columbia University before this date.

The Committee requires that applicants should be qualified by proper geological training and experience to undertake the investigation of some problem in or related to Economic Geology. Each candidate is expected to submit with his application a definite statement of the problem which he proposes to study. The carrying out of the investigation will be under the oversight of the committee and may be undertaken at any place or institution which may be preferred by the holder of the fellowship and which will meet the approval of the committee. The place and publication of results will be decided by the committee. The committee will require that the holder of the fellowship agree to give his entire time and energies to the problem selected, and further agree to contract no other engagements conflicting with or restricting this work without its consent. No objection will be made to the use of the results as a dissertation for the degree of Ph. D. in an approved university.

BOARD OF DIRECTORS

Meeting of Nov. 20, 1914.—Charles F. Rand was unanimously elected as the representative of the Institute on the John Fritz Medal Board of Award.

E. Gybbon Spilsbury was unanimously elected to succeed himself in January, 1915, on the Library Board of the United Engineering Society.

The President appointed Messrs. Walter Douglas, Charles E. Mills, and L. D. Ricketts as delegates to the seventeenth National Convention of the American Mining Congress, Phoenix, Ariz., Dec. 7 to 11, 1914.

It was voted that the sum of \$857.61, or so much of it as may be necessary, be advanced to the Land Fund from the general funds of the Institute, in anticipation of the collection of unpaid installments of subscriptions to the Land Fund; and that the Treasurer be instructed to pay to Andrew Carnegie at the present time the sum of \$2,500 on account of the Land Fund indebtedness.

It was voted to begin the San Francisco meeting on Sept. 22, 1915, none of the technical sessions to run beyond Sept. 25, 1915.

A committee was appointed to report at the next meeting of the Board as to the desirability of a committee being appointed to investigate and report on the relative merits of turbo blowers and the reciprocating engine for blowing iron blast furnaces.

Upon the recommendation of the Committee on Membership, 65 Members, 4 Associates, 2 Juniors were elected and the status of one Associate was changed to that of Member.

The resignation of one member was accepted.

AFFILIATED STUDENT SOCIETY NOTES

The Student Branch of the American Institute of Mining Engineers at the Iowa State College, Ames, Iowa, has elected officers as follows for the ensuing year: President, Charles C. Stevens; Secretary-Treasurer, W. C. Bean.

The officers of the State College of Washington Mining and Geological Society for the year 1915 are: President, George Forrest; Secretary, Aubrey Miller.

The Pick and Hammer Club of the University of Oklahoma, Norman, Okla. has made application to be enrolled as an Affiliated Student Society. The officers of the Club are: Roy A. Hazeltine, President; Evert Woolsey, Vice-President; and Donald E. Walker, Secretary-Treasurer.

The Mining Society of the Pennsylvania State College has elected officers as follows for the academic year 1914-1915; President, Robert M. Hutchison; Vice-President, John D. Cooner; Treasurer, Edward C. Brigham; Secretary, Guy C. Faust.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Nov. 10 to Dec. 10, 1914:

H. F. Grondijs, Oruro, Bolivia.
R. W. Shumway, Denver, Colo.
J. N. Houser, Mascot, Tenn.
F. L. Clerc, Denver, Colo.
Robert Sticht, Queenstown, Tasmania.
Francis C. Phillips, Pittsburgh, Pa.

E. B. Kirby, St. Louis, Mo.
W. R. Webster, Philadelphia, Pa.
F. S. Titsworth, Denver, Colo.
Frank R. Wicks, Butte, Mont.
C. Q. Schlereth, Denver, Colo.

Robert Hallowell Richards was given a dinner on December 7 at Boston by the corporation, faculty and alumni of Massachusetts Institute of Technology in recognition of his 50 years as a devoted alumnus and teacher and as an expression of the respect and affection in which he is held.

Arthur Crowfoot, who for the past 19 years has been associated with the Boston and Montana Reduction Department of the Anaconda Copper Mining Co., at Great Falls, Mont., has been appointed Concentrator Superintendent for the Arizona Copper Co., Ltd., with headquarters at Morenci, Ariz., the appointment to take effect Jan. 1, 1915. Since the closing down of the Great Falls plant, Mr. Crowfoot has been employed at the Washoe Reduction Works of the Anaconda Copper Mining Co., at Anaconda, Mont.

Frank H. Fovargue and **Ruth Hull** were married on Nov. 17, 1914, at Willoughby, Ohio. Mr. and Mrs. Fovargue will make their home at Terlingua, Texas.

W. B. Rhodes has accepted a position with the Oneida Stagg Mining & Milling Co., Idaho Springs, Colo.

Adolphe Lewisohn, of New York, has been chosen President of the Kerr Lake Mining Co.

Edward Keller, for 22 years the metallurgical representative of the Anaconda Copper Mining Co. at Perth Amboy, N. J., has been placed in charge of the sampling and chemical departments of the Raritan Copper Works, which has been consolidated with the Anaconda Copper Mining Co.

Dr. T. Poole Maynard, senior member of the associated geological, mining and civil engineers, Maynard-Carter-Simmons, will take up his residence in Atlanta, Ga., after Jan. 1. Dr. Maynard will have charge of the Atlanta office, while Messrs. Carter and Simmons will remain in the Chattanooga office.

Everett W. Smith has accepted the position of assayer for the Weedon Mining Co., Ltd., of Weedon, Que., Canada.

N. Truschkoff, recently at the Eiderlinsk mines in the department of Orenburg, is now manager of the Ekibastous mines and smelting plant of the Khirgis Mining Co., Ltd., in the Semipalatinsk district, Siberia.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, aged 40, member Institute of Mining and Metallurgy 16 years, with varied experience. Actual underground work, foreman, surveying, milling, mine examination, superintendent, manager, and other positions of trust. Several years' European practice. Familiar with flotation process. Desires responsible position. References furnished. No. 172.

Member, aged 40, graduate Massachusetts Institute of Technology. Twenty years' successful experience as blast-furnace superintendent (merchant and steel works), covering operating, remodeling and building. General superintendent and general manager of proposition comprising blast furnaces, coal and coke, and iron-ore operation, etc. Good organizer and handler of men. Desires position in above line anywhere. No. 173.

Member, technical graduate, aged 43, with 20 years' experience in management of gold, silver, and copper properties and also coal and smelting properties in United States, Canada and Mexico. Owing to illness compelled to resign. Now open for engagement. No. 174.

Member, technical graduate, with many years' experience with large smelting and refining companies as chemist, metallurgist, and superintendent, desires position. No. 175.

Member, aged 34, married, technical graduate, ten years' experience as mining engineer and mine superintendent, desires position. Can show low cost records and good references. No. 176.

Member, graduate engineer, aged 44, with over 20 years' experience in connection with large coal and coke operations in different parts of the country, desires position as general superintendent of large coal company, or will take an interest with the management in a smaller company. No. 177.

Member, technical graduate, aged 31, with extensive experience as engineer and geologist for silver mining, petroleum and dredging companies, open for engagement. Speaks Spanish fluently. No. 178.

Member, technical graduate, aged 29, with experience as engineer and superintendent for coal companies in Pennsylvania and West Virginia, desires similar position. No. 179.

Member, recent graduate, aged 27, with experience as geologist, engineer and chemist for a zinc company, desires position. No. 180.

Technical graduate, aged 30, married, specializing in chemistry and civil engineering, with nine years' experience in copper smelting, converting, electrolytic refining and reverberatory practice, three years of which was in capacity of assistant superintendent of a copper plant. No. 181.

Member, aged 41, technical graduate, 15 years' experience in mining and milling as engineer, executive head of mining and milling and as manager on large properties, desires position as superintendent, manager, or field engineer. Willing to go anywhere. Best of references. No. 182.

Member, with 18 years' experience in Western States and Latin America, desires to correspond with capitalist to finance prospecting trip. No. 183.

REPORT OF THE COMMITTEE ON MINING LAW

*Committee on Mining Law*HORACE V. WINCHELL, *Chairman*CORNELIUS F. KELLEY, *Vice-Chairman*. CURTIS H. LINDLEY, *Vice-Chairman*.

JOHN W. FINCH, <i>Secretary</i> , 730 Symes Bldg., Denver, Colo.	Frederick T. Greene	William Scallon
Albert Burch	Joseph A. Holmes	C. H. Shamel
J. Murray Clark	Edwin O. Holter	Frank L. Sizer
Will L. Clark	Edmund B. Kirby	Joel F. Vaile
C. Lorimer Colburn	Mark L. Requa	Walter H. Wiley
Courtenay DeKalb	George W. Riter	
Charles W. Goodale		

Nov. 13, 1914.

MR. B. B. THAYER, *President*,
American Institute of Mining Engineers.

Dear Sir:

Your Committee on Mining Law, through its Chairman, submits the following report:

The members of the Committee prepared a series of articles upon different phases of the mining law, which were presented during the year.

Finding that the attention of Congress was being called to the importance of mining-law revision and that bills had already been introduced affecting certain portions of the mining law and providing for the establishment of a commission to consider the subject and make recommendations, some three thousand letters were addressed by the Chairman of this Committee to mining men in the United States and Canada, asking them to communicate with their representatives in Congress and urge upon them the importance of the matter and the passage of the desired legislation. Replies were received giving notice of the writing of more than three hundred letters, in compliance with the suggestions contained in the circular letter referred to. It is probable that as many more letters were written of which no notice was given to the undersigned.

In February last, at the request of the present and past Presidents of the Institute, the undersigned visited Washington and appeared before the Senate Committee on Mines and Mining, urging upon them the passage of the act creating a commission and calling their attention to the importance of mining and the necessity for governmental encouragement and recognition. In this matter we received the cordial aid of Dr. J. A. Holmes, the Director of the Bureau of Mines and a member of this Committee.

Upon a later occasion, also at the suggestion of the President of the Institute, another committee, consisting of Messrs. W. L. Saunders, T. T. Read, and Hennen Jennings, if I am correctly informed, visited Washington and appeared before committees of Congress in connection with this matter.

The Smoot bill for the creation of the commission passed the Senate, but was never considered in the House. The Taylor bill in the House was recommended for passage, with certain amendments, but remains as unfinished business for the next session.

It developed in the course of our work that the wording of our constitution made it improper for the Institute to take action as a body and to make official recommendations to Congress expressing the desires of its members. This limitation of the constitution was a serious handicap upon the Committee's labors and efficiency. It made it impossible for

us to urge the matter before Congress and give the assurance that our recommendations were authorized by the largest body of mining engineers in this country.

The Committee feels that it has performed a useful work in the way of education and of agitation. It has presented in concrete form some of the objections to and defects in the present mining laws. It has made clearer to its members and to the public at large, so far as their attention has been directed to the matter, the reasons for our revision campaign, and some of its expressions have been used by members of Congress in calling attention to the matter there.

There will, no doubt, be further agitation of the subject before the next session of Congress and it is quite to be expected that further legislation will be enacted.

We desire to urge upon the members of the Institute the importance of the subject and the necessity for constant attention to the proceedings of Congress. If our constitution is amended so as to make it possible, we should immediately endeavor to elicit an expression of indorsement from the membership of the Institute in such clear-cut language that it could be presented to Congress for its consideration.

The Committee has not found it possible to procure a discussion of every topic upon its program. It has felt that although the subject of uniformity of mining regulations in various States might be a part of its field of consideration, yet that its hands were full at the present time in the endeavor to procure action upon the part of our Federal Legislature in connection with the mineral land laws. It has been successful in providing for co-operation between the Committees on Mining Law of the Mining Congress, the Mining and Metallurgical Society, and the Institute, but, as before stated, has been unable to take the part to which it is entitled because of the prohibitions of our constitution.

It is not within the scope of this Committee's labors to provide an endless flow of contributions to our *Transactions*. We would like to accomplish something in the direction of mining-law revision, and we earnestly hope that our constitution will be so amended as to make it possible for us to do so.

Respectfully submitted,
HORACE V. WINCHELL, *Chairman*.

BOSTON LOCAL SECTION

Executive Committee

HENRY L. SMYTH, *Chairman*

ALFRED C. LANE, *Vice-Chairman*

AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.

ROBERT H. RICHARDS

ALBERT SAUVEUR

Meets on the first Monday evening of each winter month

Twenty-third Meeting

The Twenty-third Meeting of the Boston Local Section was held at the Engineers' Club, 2 Commonwealth Ave., on Monday evening, Nov 30, 1914. Prof. R. A. Daly addressed the meeting on The Iron Ores of Kiruna, Sweden.

AUGUSTUS H. EUSTIS *Secretary*.

PUGET SOUND LOCAL SECTION

Executive Committee

I. F. LAUCKS, *Chairman*

J. F. MENZIES, *Vice-Chairman*

GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.

W. C. BUTLER

H. L. MANLEY

Meets on second Saturday of each month

The Puget Sound Section of the A. I. M. E. held its annual meeting Saturday, Nov. 14, at the College Club, Seattle. At this meeting the following Executive Committee was elected.

Chairman, I. F. LAUCKS, Seattle

Vice-Chairman, J. F. MENZIES, Carbonado

Secretary-Treasurer, GLENVILLE A. COLLINS, Seattle

[*Members of Executive Board*, W. C. BUTLER, Everett; H. L. MANLEY, Seattle

Following a dinner, the meeting was addressed by J. C. McAllen, on the Willow Creek Mining District of Alaska, and by Prof. Milnor Roberts, on a recent trip to the Queen Charlotte Islands, B. C. A lively discussion of mining developments in Alaska followed the addresses.

GLENVILLE A. COLLINS, *Secretary*.

COLUMBIA LOCAL SECTION

Executive Committee

FRANK A. ROSS, *Chairman*

RUSH J. WHITE, *Vice-Chairman*

L. P. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.

FRANCIS A. THOMSON

FREDERIC KEFFER

The fourth annual meeting of the Columbia Local Section of the American Institute of Mining Engineers was held in the Spokane Hotel, Spokane, Wash., Nov. 13, 1914. Chairman Francis A. Thomson presided and about 45 members and guests were present. Short addresses on the general mining situation were given by John B. Fiskien, Lorne A. Campbell, Oscar Lachmund, Frederic Keffer, George H. Wyman, Jr., S. H. Richardson, and Prof. D. C. Livingston.

The Executive Committee for the year 1915 was elected, as follows:

FRANK A. ROSS, *Chairman*

RUSH J. WHITE, *Vice-Chairman*

L. P. ARMSTRONG, *Secretary-Treasurer*

FRANCIS A. THOMSON

FREDERIC KEFFER

Prof. F. M. Handy read a paper on The Economic Geology of Eastern Washington.

L. K. ARMSTRONG. *Secretary*.

MOTION PICTURES AND LANTERN SLIDES OF U. S. BUREAU OF MINES

The following described motion-picture film and lantern slides will, on application to the U. S. Bureau of Mines, Pittsburgh, Pa., be loaned or shown by the Bureau under the following conditions:

1. They will be loaned or shown, preferably at meetings of miners, mine operators, mine inspectors, mining institutes, associations or similar gathering of persons concerned in the production, manufacture, or utilization of any of the minerals of the United States. They may, however, be loaned for general meetings other than those listed above, in which safety or prevention of waste in any industries are important factors.

2. They will not be loaned or shown at meetings of the general public excepting where the development of an interest in mining problems and in safety in the mining or metallurgical industries is an important factor.

3. In no case will these film and lantern slides be loaned or shown at public meetings where entrance charges are made.

4. They will be loaned preferably only on application of responsible persons concerned in the mining or metallurgical industries. Responsible persons engaged in other industries may borrow them when the purpose is to promote safety or prevent waste.

5. They will be loaned only providing the applicant guarantees that they will be shown by an experienced motion-picture or lantern-slide operator, and providing further that he pays expressage both ways and guarantees or insures the safe return of the same, at an appraised value of \$150 per reel of film, and 25c. per lantern slide.

6. On especial occasions, upon application, the Bureau may provide an engineer or other lecturer, and, on exceptional occasions, its own projecting machine and operator. The Bureau will be glad to receive suggestions to this effect.

7. Deviations from the above rules can be made only by authority of the Director of the Bureau of Mines.

MOTION PICTURE FILM

The Bureau of Mines possesses duplicates of subjects 2, 4, 14, and 20 only. It possesses the negatives of subjects 8, 10, 11, 12, 13, 14, 15, 16, 17, and 18. Only the above subjects will be loaned for showing by others than employees of the Bureau.

Subject 1: Operations of the Oliver Iron Mining Co.

- Reel 1. Stripping Operations.
- Reel 2. Exploration and Drilling.
- Reel 3. Underground Mining.
- Reel 4. Logging Operations.

Subject 2: Mine Explosion and Rescue at the Lick Run Mine of the Pittsburgh Coal Co., Bruceton, Pa.

Showing scene at mine, followed by an explosion; telegram is then received at the Bureau of Mines telling of disaster, and rescue car is immediately dispatched; rescue men enter mine, bring out injured and apply artificial respiration.

Subject 3. An American in the Making.

Showing a European peasant receiving passage money to go to America; on arrival is met by his brother and taken to Gary, Ind.; is guided to Y. M. C. A. instead of saloons to spend his leisure hours; unfitted for skilled labor, is given employment as a

laborer and his first lesson is one of safety; he is shown goggles to protect the eyes from flying chips, a safety hook, a hand protector with safety-spring wrist band, a guard for gauge glass, a locking switch, and universal danger sign. Transferred to Lorain, Ohio, he learns of other safety appliances; for example, guards on drill press, guard on planer, guard on circular saw, chip guard on shaper for protection of eyes, guards on emery wheel, guard over belt and pulleys and guards over feed gears of lathe. He returns to Gary and goes to work at open-hearth furnaces and converter. Teacher in night school becomes interested in workman, they are married, and six years later, happy family is shown with son in attendance at model school and playing on the company's playground.

Subject 4: National Mine Safety Demonstration, Pittsburgh, Oct. 31, 1911, under Auspices of U. S. Bureau of Mines, American National Red Cross, and Pittsburgh Coal Operators' Association.

Showing President Taft arriving at the demonstration, immediately after which the first-aid competition commences. Details of the treatment of electric shock are shown and details of rescue men wearing helmets also are shown. The President presses a button and causes explosion of coal dust in the experimental gallery, after which the rescue men enter the smoking gallery and bring out injured and apply first aid. The President and his party then go to speaker's stand and following speeches by the President, Secretary Fisher, and Miss Boardman (of the American Red Cross), trophies are presented to all competing in the meet.

Subject 5: The Miner at Work, at the Marianna Mine of the Pittsburgh-Buffalo Co.

Showing miners descending into the mine, followed by view of the hoisting engineer at work; mine cars are distributed to the working places underground; mine foreman giving instructions in his underground office; miner is shown at his working place; loaded trip is made up and taken to the shaft bottom by compressed-air locomotives; loaded mine car sent to surface; miners are sent to the surface, where they leave their safety lamps in lamp room.

Subject 6: Manufacture of Coke, at the Marianna Mine of the Pittsburgh-Buffalo Co.

Coal for coking is cleaned and refuse disposed of; larry takes coal to row of coke ovens to point where an oven is charged and then leveled. After 72 hr., the ovens are opened and coke is quenched and drawn.

Subject 7: Mine to Molder (Iron). (3 reels.)

Showing iron-mining and blast-furnace operations and steel plants of Rogers, Brown & Co.; stripping operations; ore train arriving at docks; unloading ore train into ore boats; shipping canal at Sault Ste. Marie; unloading iron ore at Buffalo; charging blast furnace; making pig iron; drawing slag; breaking up pig iron; pig-casting machine; handling pig iron with an electric magnet; Bessemer converter; pouring ingots; rolling railroad rail.

Subject 8: Demonstration of First Aid to Injured Miners.

1. An accident resulting in a dislocation of the shoulder, which is immediately reduced by injured miner's companion.

2. Miner injured underground is brought out by "one-man carry," resuscitated by Sylvester method, and burns of abdomen dressed.

3. Trolley wire is displaced by careless miner and another falls on it and is shocked. Miners trained in first aid, resuscitate by Schafer method and bandage burned hands.

4. Details of Bandaging.

(a) The eye; (b) the throat; (c) the forearm; (d) the elbow; (e) the arm and hand; (f) the upper arm; (g) the hand; (h) the top of the head, etc.

Subject 9: Demonstration of Inflammability of Coal Dust and Use of Rescue Apparatus at the Gas and Dust Gallery No. 1, Bureau of Mines Testing Station, Pittsburgh, Pa.

Showing explosion of coal dust by ignition of black blasting powder, followed by the entry of rescue men into the smoke gallery, returning with injured men, resuscitating them and applying first aid.

Subject 10: Transportation of Coal.

Trips of loaded mine cars come from mine portal, go to tippie, and are dumped. Next, coal is loaded into coal cars from tippie, several different methods being shown.

A long train of coal cars *en route* to the dock is followed by scenes showing various methods of loading into the boats.

After the voyage, the ship is unloaded. Coal is transported to distributing points and retailed.

Subject 11: Welfare Work in Mining and Allied Industries.

Showing various features of welfare and social improvement work which have been undertaken by various concerns; e.g., swimming pools, playgrounds, hospitals, change and wash houses, etc.

Subject 12: Safe Method of Bituminous-coal Mining at the U. S. Coal & Coke Co. Mines, Gary, W. Va. (2 reels).

Showing assistant foreman testing for gas and inspecting roof, machine runner at work; miners loading coal; miners laying track; drilling for shooting; charging explosive; loading coal following shot; miners posting places; trip of loaded cars coming from mine; empties returned; dumping in tipples, loading coal into railroad cars; miners quitting work, leaving checks and returning home.

Subject 13: Anthracite-coal Mining (2 reels).

Stripping operations at Harleigh, Pa., and Lattimer, Pa.; underground mining at Nanticoke, Pa.; also surface operations which include reclaiming a culm bank, breaker operations, loading and shipping at Nanticoke.

Subject 14: Joint Field Meet of U. S. Bureau of Mines and American Mine Safety Association, Pittsburgh, Pa., September, 1913.

Showing mine-rescue teams contesting for prizes; rescue of miner overcome by afterdamp and a miner who has erected a barricade to the afterdamp; explosion at the Experimental Mine of the Bureau of Mines; rescue crew of the Bureau of Mines motoring to scene, rescue men donning apparatus and entering mine.

Subject 15: Blast of 8 1/2 Tons of Powder at Limestone Quarry of Bethlehem Steel Co., McFee, N. J.

Subject 16: Results of Explosion Tests at Experimental Mine.

Subject 17: Blast of 40 Tons of Powder at Quarries of Tompkins Cove Stone Co., Tompkins Cove, N. Y.

Subject 18: Mining Magnetic Iron Ore at the Mines of Witherbee, Sherman Co., Mineville, N. Y. Safe Methods.

Shows the details of mining iron ore, including drilling, blasting, hauling, hoisting, timbering, and the details of separating and shipping the ore; the arrangement of the plant and town, and details of power plant.

The whole picture is arranged with a view to emphasizing and promoting safety.

Subject 19: Marble Industry.

Details of channeling, drilling, and blasting out blocks of stone in the quarry.

Finishing plant, showing operation of saws, planers, lathes, and buffers, and showing sculptors at work.

Subject 20: Granite Industry.

Details of blasting, wedging, and hoisting stone from the pit.

Details of rough shaping, polishing, turning, and carving.

Subject 21: The Shooting of the Lake View Gusher; General View of Oil Fields; Drilling Well.

The shot followed by tremendous flow of oil.

Rear view of gusher showing surging oil, flowing through ditches and finally impounded in a reservoir.

Subject 22: From Blast Furnace to Finished "National" Pipe.

Showing ladle cars with molten crude iron from blast furnace taking charge to open-hearth steel furnaces and Bessemer converter; after the heat furnaces are tapped and ingots poured and stripped; blooming mills and various rolling processes prior to pipe welding; details of equipment and operation used in making "butt weld" pipe.

Subject 23: The Man Who Learned. (Sanitary.)

Will protests to his father-in-law about his unsanitary methods; the milker with unwashed hands—"untidy ways mean unhealthy milk"; obstinate even though it was the means of separation from his little grandson; the baby critically ill, impure milk the cause; his own grandson dying from the effects of impure milk, one of the thousands of such cases; some time later the convalescent receives an invitation; just arrives when the first improvement starts; the new pastures; the sanitary barns; cleaning the cows before milking, vacuum cleaning, brushing, spraying and washing; sending the milk to be bottled; bottling by machinery, no hand touches the milk; a reunited family.

Subject 24: The Fly Pest.

Lying eggs in outside meat; the eggs attacked by Ichneumon fly; the hatching of the eggs; maggots one hour old; maggots seven days old; entering earth to become pupal; the pupa stage one day later; the fly emerging wingless from the earth; eleventh day, the fly full grown; fly taking syrup from a needle point; fly's tongue; foot of a fly; how flies carry contagion; how the fly spreads tuberculosis.

Subject 25: War on the Mosquito.

Examination of the pool to find if larvæ are breeding; mosquito eggs hatching in the water; mosquito larvæ—they breathe through their tails; mosquito in intermediate or pupa stage; they breathe through their horns; mature mosquito emerging from pupa; treatment of the pool with oil—the oil prevents the mosquito larvæ from breathing;

cleaning out a typical breeding place before filling with dirt; treatment of another place with oil; tin cans holding stagnant water barred; mosquitoes by the millions; health officer teaching school children to aid him in his work; larvæ water beetle devouring larvæ of mosquitoes; mosquito biting a man's hand—notice the body filling with blood.

LANTERN SLIDES

Grouped by subjects with number under each.

Subject A. General For the Mining Industry	
1. First Aid to the Injured.....	50
These slides are also prepared with titles in English, German, Italian, and Polish on each, so that they can be used before an audience of mixed foreigners without any other accompanying lecture.	
2. Mine Rescue and Relief.....	35
3. Welfare.....	25
4. Testing and Use of Explosives.....	40
5. Safety.....	35
6. Statistics relating to Production and Accidents.....	15
Subject B. Coal Mining	
1. Mining Methods relating to Use of Explosives, Timber, etc. ...	25
2. Structures, Tipples, Breakers, etc.....	20
3. Equipment, Drills, Cars, Locomotives, Cutters, etc.....	15
4. Accidents, Disasters, etc.....	25
5. Statistics, Accidents.....	20
6. Statistics, Production.....	10
7. Dangerous Practices.....	100
8. Experimental Mine of the Bureau.....	35
9. Coke Industry.....	25
Subject C. Use of Coal	
1. Briquetting.....	20
2. Steam Power Plants Including Smoke Prevention.....	30
3. Producer Gas Power Plants.....	15
4. Coal Handling.....	15
Subject D. Petroleum and Natural Gas	
Scenes in Oil Fields of Derricks, Burning Wells, Gushers, Natural Gas-Gasoline Plants, etc.....	50
Subject E. Iron and Steel	
1. Metallurgy of Iron and Steel.....	20
2. Rolling Mills.....	20
3. Surface Structures at Iron Mines.....	10
Subject F. Gold Mining	
Surface Structures and Equipment.....	20
Subject G. Rare Metals	
Geology, Outcrops, etc.....	25
Subject H. Copper Mining	
Structures, Equipment and Methods.....	25
Subject I. Stone Quarrying	
Structures, Equipment and Methods.....	50

In addition to the films in the foregoing list, the following are in course of preparation:

The Miner's Lesson—showing how a careless miner, who disregarded regulations, danger signs, etc., came to grief.

Smelting of Zinc.

Indiana Limestone Industry.

Marquette Cement—From Quarry to User.

Bituminous Coal Stripping, Pittsburgh, Kan.

Lead Mining at Joplin, Mo.

Work of the Radium Institute in the Paradox Valley and Denver, Colo.

Mining and Smelting of Copper at Bingham Canyon, Utah, and Ely, Nev.

LIBRARY

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AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining

- MODERN TUNNELING, WITH SPECIAL REFERENCE TO MINE AND WATER SUPPLY TUNNELS. By D. W. Brunton and J. A. Davis. New York, 1914.
- MAPS OF THE MINING DISTRICTS OF MINNESOTA, 1914. Minnesota School of Mines, University of Minnesota.
- GREAT COLLIERY EXPLOSIONS AND THEIR MEANS OF PREVENTION. By W. Galloway. London, n. d. (Gift of Author.)
- PREVENTION OF ACCIDENTS FROM EXPLOSIVES IN METAL MINING. Miners' Circular 19, U. S. Bureau of Mines. Washington, 1914.
- ROCK EXCAVATING AND BLASTING. By J. J. Cosgrove. Pittsburgh, 1913.
- BERGBAU UND HÜTTENWESEN. By E. Treptow, S. Wülf, and W. Borchers. Leipzig, 1900.

Metallurgy

- METALLURGY OF THE NON-FERROUS METALS. By William Gowland. London, 1914.
 DIE LEGIERUNGEN IN IHRER ANWENDUNG FÜR GEWERBLICHE ZWECKE. Ed. 4. By A. Ledebur. Berlin, 1913.
 GIESSERIE-MASCHINEN. By Emil Treiber. Leipzig, 1911.
 HANDBUCH DER GATTIERUNGSKUNDE FÜR EISENGIESSERIEEN. By R. Weber.
 AMERIKANISCHE GIESSERIE-PRAXIS. By Thomas D. West. Berlin, 1910.
 GRUNDZÜGE DER WALZENKALIBRIERUNG. By Emil Kirchberg. Dortmund, 1905.
 DAS HARTEN DES STAHLER IN THEORIE UND PRAXIS. Ed. 6. By F. Reiser. Leipzig, 1913.
 EISEN UND STAHL. Ed. 2. By Alois Wurm. München, 1911.
 LES ACIERS SPÉCIAUX. By Léon Guillet, preface by Henry Le Chatelier. Paris, 1904.
 DIE ELEKTROLYTISCHEN METALLNIEDERSCHLÄGE. By W. Pfannhauser, Jr. Berlin, 1910.

Chemistry and Metallography

- DIE CHEMIE DES EISENS. By Friedrich Toldt. Leoben, 1898.
 METALLOGRAPHIE IN ELEMENTARER DARSTELLUNG. By Rudolf Ruer. Hamburg u. Leipzig, 1907.
 DIE METALLHÜTTENCHEMIE. By Max Orthey. Leipzig, 1910.

General

- UNIT CONSTRUCTION COSTS FROM THE NEW SMELTER OF THE ARIZONA COPPER CO., LTD. By E. Horton Jones. New York, 1914.
 DIE FRAGE DER SELBSTKOSTENBERECHNUNG VON GUSSSTÜCKEN IN THEORIE UND PRAXIS. By Engelbert Leber. Düsseldorf, 1910.
 PRIMER OF SCIENTIFIC MANAGEMENT. Ed. 2. By Frank B. Gilbreth. New York, 1914. (Gift of Author.)
 [Note.—The new edition is issued a little more than two years after the old one, showing a constant demand for an elementary treatise on the subject.—W. P. C.]
 DIE DEUTSCHE EISENINDUSTRIE IHRE GRUNDLAGEN, IHRE ORGANISATION UND IHRE POLITIK. By Erhard Hübener. Leipzig, 1913.
 GEWICHTSTABELLEN FÜR WALZEISEN. Ed. 6. By R. Ziebarth. Berlin, 1909.
 DIE SCHULE DES WERKZEUGMACHERS. Ed. 3. By Fritz Schön. Hannover, 1911.
 MECHANISCHE TECHNOLOGIE. By Schimpke. Leipzig, 1912.

Company Reports

- ENGINEERS' REPORT ON THE GOLD PLACERS OF THE MARANON OF THE PERUVIAN EXPLORATION CO. By Raymond McCune. (Gift of Kirby Thomas.)
 SENECA-SUPERIOR SILVER MINES, LTD. Second Annual Report. 1914. (Gift of Kirby Thomas.)
 UNITED CHEMICAL AND NICKEL CORPORATION. Description of. Boston, n. d. (Gift of Corporation.)
 UNITED STATES STEEL CORPORATION. Annual Report, 6th-12th, 1907-1913. Hoboken, N. J., 1907-13. (Gift of U. S. Steel Corporation.)

Trade Catalogues

- CHICAGO PNEUMATIC TOOL CO., Chicago, Ill. Bulletin 34-S. Small power-driven compressors. Nov., 1914.
 DETROIT FUSE & MANUFACTURING CO., Detroit, Mich.
 "Detroit" ironclad fused switches and motor starters.
 "Arkless" indicating enclosed fuses. 28 pp.
 EXCELSIOR DRILL & MFG. CO. Bulletin A2. Excelsior airometer.
 GENERAL ELECTRIC CO., Schenectady, N. Y.
 Bulletin 44003. Modern electric railway apparatus. Oct., 1914.
 Bulletin 3317. Cords and portables, N. E. code. Oct., 1914.
 GOLDEN ANDERSON VALVE SPECIALTY CO., Pittsburgh, Pa. Double-cushioned triple acting and non-return valves.
 HERBERT & HUESGEN CO., New York, N. Y. New Spencer Delineascope, Model 8.
 INGERSOLL-RAND CO. New York, N. Y. Form 632. A few remarks about Calyx core drills.
 NATIONAL LEAD CO., New York, N. Y. The Cinch expansion bolt and Cinch anchoring specialties. 23 pp.
 PERCY PITMAN, Westminster, London, England. Modern Pelton wheels and governors.
 STEPHENS-ADAMSON MFG. CO., Aurora, Ill. Labor Saver. No. 66.
 WEBSTER MFG. CO., Tiffin, Ohio. Webster Method. Nov., 1914.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Nov. 10 to Dec. 10, 1914:

Members

- ABBOTT, LOUIS BENJAMIN, Chief Engr., Consolidation Coal Co., Elkhorn Div.,
Jenkins, Ky.
- ANDERSON, THORVALD J.....542 S. Seventh Ave., Mt. Vernon, N. Y.
- BALFOUR, A. M.....P. O. Box 301, Republic, Wash.
- BOUDWIN, WALKER J., Mech. Engr.,.....159 Pierpont Ave., Salt Lake City, Utah.
- BRADLEY, LINN, Engr.....Research Corpn., 63 Wall St., New York, N. Y.
- BROKAW, ALBERT DUDLEY, Asst. Prof., Min. and Econ. Geol., Univ. of Chicago,
Chicago, Ill.
- BUTLER, PERCIVAL PAGE, Asst. Supt., Copper Queen Reduction Wks., Douglas, Ariz.
- CAJANDER, SVEN G., Mech. Engr.....Braden Copper Co., Rancagua, Chile.
- CLARKE, EDGAR W., Chief Engr.....Cambria Steel Co., Johnstown, Pa.
- CRANE, CLINTON W., Pres., St. Joseph Lead Co., 61 Broadway, New York, N. Y.
- CREGAN, JOHN F., Genl. Supt.....Embree Iron Co., Embreeville, Tenn.
- DAVIS, JOHN HASKELL, Min. Engr., Tombstone Consolidated Mines, Tombstone, Ariz.
- EDELEANU, LAZAR, Chem.....Unter den Linden 35, Berlin W. 8, Germany.
- EVANS, CADWALLADER, JR., Genl. Mgr., Acadia Coal Co., Ltd.,
Stellarton, N. S., Canada.
- GARNSEY, CYRUS, JR., Coal Operator.....Galloway Coal Co.,
Memphis, Tenn.
- GATES, LAUREN A., Div. Engr.,Four States Coal & Coke Co., Dorothy, W. Va.
- GIOLITTI, FEDERICO, Prof. of Met.....Corso Moncalieri 75, Torino, Italy.
- GREENE, FRANK CASANOVE, Consulting Engr. and Coal Geol., 1126 Harvard Ave.,
Seattle, Wash.
- GREENE, OSCAR VERNON, Min. Engr. and Coal Geol., 1405 Scofield Bldg.,
Cleveland, Ohio.
- GUITERMAN, KENNETH A., Chief Chem., Research Lab., Amer. Smelt. & Ref. Co.,
Maurer, N. J.
- HAFF, RAYMOND E. T.....Room 329, Penna. Station, New York, N. Y.
- HENDERSON, FREDERICK BRADLEY, Asst. Genl. Mgr., Associated Oil Co.,
808 Sharon Bldg., San Francisco, Cal.
- HOTTA, MASAICHI, Min. Engr.....Fujita & Co., Osaka, Japan.
- HUGHES, HOWARD R., Pres.....Sharp-Hughes Tool Co., Houston, Texas.
- JENSEN, JOSEPH, Mineral Inspector, U. S. General Land Office, 512 Custom House,
San Francisco, Cal.
- KANARR, HARRY M., Chief Engr., Punxsutawney Div. Rochester-Pittsburgh
Coal & Iron Co., Punxsutawney, Pa.
- KYNOR, HERBERT D., Asst. Dist. Supt., Lehigh Coal & Navigation Co., Lansford, Pa.
- LATHE, FRANK E., Lecturer in Met., Mining Building, Univ. of Toronto,
Toronto, Canada.
- LAUCHLI, EUGENE HENRI, Civ. Engr.....Tallulah Falls, Ga.
- LEWIS, JOHN MOORE, Supt.....Thacker Coal & Coke Co., Thacker, W. Va.
- LOYD, BERT, Coal Mine Supt.....Sugarite, N. M.
- MCLEAN, JAMES, Vice-Pres., Phelps, Dodge & Co., 99 John St., New York, N. Y.
- MICKEL, ROBERT ELI, Min. Engr., Staff Quarters, Crown Mines,
Johannesburg, S. Africa.
- MAGAZUMI, JUNJIRO, Prof. of Mining, Kiushu Imperial Univ., Fukuoka, Japan.
- O'NEALE, M. L., Supt.....Coal City Mining Corpn., Coal City, Ala.
- PACK, FREDERICK J., Deseret Prof. of Geol., Univ. of Utah, Salt Lake City, Utah.
- PROUTE, CHARLES YALE, Student.....1136 E. 1st South, Salt Lake City, Utah.
- PRESCOTT, BASIL, Min. Geol.....711 Mills Bldg., El Paso, Texas.
- ROBERTSON, FREDERICK YOUNG, Genl. Mgr. and Vice-Pres., U. S. Metals
Refining Co., 42 Broadway, New York, N. Y.
- SAVAGE, LESLIE L., Mining Operator.....530 West End Ave., New York, N. Y.
- SEARLS, FRED, JR., Geol.....604 Mills Bldg., San Francisco, Cal.

SHELBY, WILLIAM H., Min. Engr., Moctezuma Copper Co., Pilares de Nacosari,
Son., Mexico.
SPEARMAN, CHARLES, Genl. Mgr., Kirkland Goldfields, Ltd.,
Kirkland Lake, Ont., Canada.
STRAHLER, ALLEN P., Min. Engr., Mogollon, N. M.
TEUSCOTT, SAMUEL JOHN, Asst. Prof., Royal School of Mines,
London, N. W., England.
WATTS, JAMES, Chief Mine Agent, St. John del Rey Min. Co., Ltd.,
Villa Nova de Lima, Minas, Brasil.
WINSOR, FRANK H., JR., 2940 Broadway, New York, N. Y.
WOOD, GEORGE REID, Div. Engr., Lehigh Valley Coal Co., Centralia, Pa.

Associate Members

HAYDEN, CHARLES, 25 Broad Street, New York, N. Y.
ROSENBLATT, GERARD B., Elec. Engr., 1212 Walker Bank Bldg.,
Salt Lake City, Utah.

[CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Nov. 10 to Dec. 10, 1914, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Amil Abner Anderson, Lead, S. D.
Proposed by Allan J. Clark, W. J. Sharwood, B. C. Yates.
Born 1890, Capula, S. D. 1913, So. Dakota School of Mines; B. S. 1908 and 1910, each, 3 1/2 months, Homestake Mine. 1911, Montezuma Mill, Colo. 1912, Homestake Mine.
Present position: 1913 to date, Metallurgical Dept., Homestake Mining Co.

Edwin Marcottel Ball, Birmingham, Ala.
Proposed by George G. Crawford, Frank H. Crockard, C. E. Abbott.
Born 1886, Florence, Wis. 1909, Grad., University of Wisconsin, in General Engineering; B. S. 1907, Division Engr., Ore Mines, Tenn. Coal, Iron & R. R. Co. 1909, Transitman, Ore Mines, Oliver Iron Min. Co. 1910, Construction Engr., Ore Mines, Tenn. Coal, Iron & R. R. Co.
Present position: 1912 to date, Supt., Ore Mines, Ishkooda Div., Tenn. Coal, Iron & R. R. Co.

Charles A. Boln, Philadelphia, Pa.
Proposed by David T. Day, Anthony F. Lucas, E. W. Parker.
Born 1876, Liverpool, England. 1883-90, General, Newton School, Rock Ferry, England. 1890-93, General, Bruges, Belgium. 1906-07, Chemistry, Drexel Institute, Philadelphia. 1905, Oil business in England. 1905-09, Oil business in America, laboratory work.
Present position: 1909 to date, oil salesman and technical work.

Alfred W. Calloway, Cumberland, Md.
Proposed by S. A. Taylor, H. H. Stoek, Floyd W. Parsons.
Born 1872, England. Grammar and high schools of Brooklyn, N. Y. 1899-1903, Engineer Corps, Penn. R. R., Bellwood Division. 1903-13, Rochester & Pittsburgh Coal & Iron Co., Punxsutawney and Indiana, Pa.
Present position: Pres., Davis Coal & Coke Co.

William Robert Calvert, Salt Lake City, Utah.

Proposed by A. F. Lucas, D. T. Day, A. H. Brooks.

Born 1878, Minden, Ontario. 1904, Univ. of Neb.; A. B. 1907, A. M., Univ. of Neb. 1902-04, Instructor Mineralogy, Univ. of Neb. 1903-04, Instructor Petrology, Univ. of Neb. 1904, Chemist, Republic Steel & Iron Co. 1905, Chemist, Tacoma Smelting Co. 1905-06, Instructor Assaying and Mining, Univ. of Wash. 1906, In charge Dept. of Geol., summer session, Univ. of Wash. 1906-07, Field Asst. 1908-09, Asst. Geologist. 1911-12, Geologist, Chairman Coal Class'n Board of the U. S. Geological Survey. 1913, Consulting Geologist, U. S. Bureau of Mines.

Present position: Consulting Geologist, private practice.

Charles Joseph Coll, Broughton, Nova Scotia.

Proposed by E. W. Parker, S. D. Warriner, Howard N. Eavenson.

Born 1864, Pittsburgh, Pa. Public and private schools, Pittsburgh and Beaver, Pa. 1887-89, Lehigh Univ., So. Bethlehem, Pa. 1883-87, Engrg. Dept., H. C. Frick Coke Co., Pa. 1889-90, Division Engr., Yough. Southern Ry., Pa. 1890-94, Division Engr., Uniontown Division, Pa. 1894-97, Division Engr., Leisenring Division, Pa. 1897-98, Supt., Mammoth Mines. 1898-1900, Supt., Lemont Mines, H. C. Frick Coke Co., Pa. 1900-12, Gen. Mgr., Acadia Coal Co., Ltd., Stelarton, N. S. 1912-13, Supt. Redstour Mines, H. C. Frick Coke Co., Va.

Present position: Genl. Mgr., Cape Breton Coal, Iron & Ry. Co., Ltd.

John Centennial Devine, Ray, Ariz.

Proposed by D. C. Jackling, Louis S. Cates, W. S. Boyd.

Born 1876, Florence, Ariz. 1891-93, St. Mary's College, Oakland, Cal. 1901-07, Asst. Supt. and General Mine Foreman, Troy Manhattan Copper Co., Troy, Ariz.

Present position: 1907 to date, Asst. Supt., Ray Consolidated Copper Co.

William A. Evans, Hartshorne, Okla.

Proposed by F. K. Copeland, Oscar F. Scholz, Carl Scholz.

Born 1873, Summit Hill, Pa. 1892, Lehigh Univ. 1896, B. S. in metallurgy. 1897-99, Engineer Dept., Lehigh Coal & Navigation Co., Lansford, Pa. 1899-1901, Constructing Engr., Montana Coal & Coke Co., Enterprise, W. Va. 1901-03, Constructing Engr., McAlester Coal Co., Hartshorne, Okla. 1903-06, Min. Engr., Otto Manuel Coal Co., Raymond City, W. Va. 1906-09, Genl. Supt., Kentucky Coal Mining Co., Morganfield, Ky.

Present position: 1909 to date, Genl. Supt., Rock Island Coal Mining Co.

Mickel Fort, Lima, Peru.

Proposed by R. S. McCaffery, Pierre Van Isschott, F. C. Fuchs.

Born 1869, Lima, Peru. 1890, Min. Engr. at the School of Engineers of Peru; Professor of Metallurgy and Ore Dressing. 1890-95, The Backus & Johnston Co., Casapalca, Peru. 1895-96, The Victoria Amalgamating Plant, Yauri, Peru. 1896-97, The Aguas Calientes, mining and dressing, Casapalca, Peru. 1897-1904, Professor of Metallurgy and Dressing Ores, Lima, Peru. 1901-04, Director, School of Engineers, Lima, Peru. 1897-1904, Consulting engineer.

Present position: Director, School of Engineers, Lima, Peru, civil, mining, metallurgical, chemical, architectural, electrical and military sections.

Lester Eames Grant, Rancagua, Chile.

Proposed by Franklin Guiterman, A. Eilers, P. A. Mosman.

Born 1884, Denver, Colo. 1902-06, Yale College, New Haven, Conn.; A. B. 1906-09, Columbia School of Mines, New York; E. M.; summer work at Leadville and Cripple Creek, Colo. 1909-10, In the concentrator of the Nevada Cons. Copper Co., McGill, Nev. 1910-11, Mine foreman and engineer, Braden Copper Co., Rancagua, Chile. 1912-13, Asst. Mill Supt., Braden Concentrator. 1913-14, At Braden Smelter.

Present position: Asst. to Genl. Mgr., Braden Copper Co.

Lester Strickland Grant, Stent, Tuolumne County, Cal.

Proposed by W. D. Waltman, H. H. Utley, Fred H. Bostwick.

Born 1877, New Haven, Conn. 1899, Engineer of Mines, Colorado School of Mines. 1899-1904, Surveyor and office man, Isabella Gold Min. Co., Cripple Creek, Colo. 1904-05, Supt., Murphy and Frye Lease, Isabella Gold Mining Co. 1905-06, Surveyor and office man, Findley Consolidated Gold Min. Co., Cripple Creek, Colo. 1906-09, Met., Inca Mining Co., Santo Domingo, Carabaya, Peru. 1909-13, Supt., Isabella Mines Co., Cripple Creek, Colo.

Present position: Genl. Mgr., Jumper Californian Gold Mines Co., Stent, Tuolumne County, Cal. Genl. Mgr., Franco Contention Mining Co., Columbia, Cal.

William J. Hamilton, Anyox, B. C., Canada.

Proposed by A. J. Bone, J. T. Dillon, W. B. Bishop.

Born 1884, Shakopee, Minn. 1902-06, Univ. of So. Calif. 1906-09, Colo. School of Mines; E. M. 1909-10, Miner, Phoenix, B. C. 1910-13, Chemist, Grand Forks, B. C.

Present position: 1913 to date, Chief Chemist and Assayer, Granby Consolidated Mining, Smelting & Power Co., Ltd.

Clarence C. Hamlin, Colorado Springs, Colo.

Proposed by Fred H. Bostwick, Richard A. Parker, D. W. Brunton.

Born 1868, Manchester, Iowa. Public Schools.

Present position: 1902 to date, Genl. Mgr., Granite Gold Mining Co., Colorado Springs, Colo.

Harvey Lockhart Handley, Yankee, New Mexico.

Proposed by T. H. O'Brien, E. B. Roeser, J. M. Clark.

Born 1885, Lewisburg, W. Va. 1892-93, Public schools. 1898-1902, Greenbrier Military Academy, W. Va. 1902-06, Washington and Lee Univ. 1903-04, Asst. Res. Engr., Construction, Chesapeake & Ohio Ry. 1906-07, Engr. and Officeman, Kentucky Coal & Lumber Co. 1907-08, Engr., Thurmond Coal Co. 1908-09, Member firm, Amick & Handley, Pikeville, Ky. 1909-10, Locating Engr., Greenbrier Cheat & Elk Ry., W. Va. 1910-11, Min. Engr., White Oak Fuel Co. 1911-13, Ch. Engr., Yankee Fuel Co.

Present position: 1913 to date, Genl. Supt., Yankee Fuel Co. and New Mexico, Colorado Coal & Min. Co.

Eugene F. Irwin, Lead, S. D.

Proposed by Allan J. Clark, W. J. Sharwood, Nathaniel Herz.

Born 1865, Clinton, Ill. Only general education in common schools of Ill., and studies at home since. Timekeeper and storekeeper for Homestake Mining Co. from April, 1893, to date, to which has been added the employing of all men hired for the past 10 years, which changed title to position held to that of "Manager of Employment Department."

Present position: 1893 to date, Mgr., Employment Dept., Homestake Mining Co

Alfred L. Johns, Arcadia, Fla.

Proposed by F. A. Ross, L. K. Armstrong, Roy H. Clarke.

Born 1886, Topeka, Kan. 1892-1904, Public schools, Colorado Springs, Colo. Junior year in High School. 1906-07, Freshman Mining Engr. Course, College. 1908-09, Sophomore Mining Engr. Course, Colo. School of Mines. 1909-10, Junior year of mining engineering course completed at Colorado College, Colorado Springs, Colo. 1904-11, Miner, Portland Gold Min. Co., Victor, Colo. 1911-12, Asst. Mine Engineer, Portland Gold Min. Co., Victor, Colo. 1912-13, Mine Engineer, Cinco Minas Co., Magdalena, Jalisco, Mexico. 1913-14, Engineer on exploration work, Cinco Minas Co.

Present position: Min. Engr., Marcus Daly Estate, New York, N. Y.

Edward Horton Jones, Clifton, Ariz.

Proposed by Norman Carmichael, John Kiddie, R. B. Earling.

Born 1878, Pontiac, Mich. 1896, Englewood High School, Chicago. 1896-98, Univ. of Mich.; B. A. 1898-99, Harvard. 1899-1901, Miner, hoisting engineer, etc., Arizona. 1901-03, Univ. of Arizona; B. S. 1903-04, Chemist and Assayer, Black Mountain Mine, Sonora, Mexico. 1904-05, Chemist, Zubiate mine, near La Colorado, Sonora, Mexico. 1905-06, Draftsman, concrete foreman, Old Dominion mine, Globe, Ariz. 1906-07, Design and placing of all concrete at Dawson coal washer, Dawson, N. M.; also charge excavation, etc. 1907-09, Contracting in Denver, reinforced concrete. 1909, Draftsman, Kelvin, Ariz. 1909-11, Construction Engr., Ray, Ariz., Ray Cons. Copper Co. 1911-12, Ray Central, Ray, Ariz.

Present position: 1912 to date, Construction Engr., Arizona Copper Co., Clifton, Ariz.

David Nelson Kay, Hayden, Ariz.

Proposed by Louis S. Cates, David D. Moffat, J. J. Ormsbee.

Born 1884, Ogden, Utah. 1890-99, Completed primary and grammar grades in public schools of Ogden, Utah. 1899-1904, Completed scientific course, Sacramento High School, Sacramento, Cal. 1904-10, Four years of the mining and geology course, Stanford Univ.; incomplete. During college vacations, underground work in following mines: 1906, Utah Copper Mine, Bingham, Utah. 1907, Eagle and Blue Bell, Eureka, Utah. 1908, Daly West, Park City, Utah. 1909, Central, Grass Valley, Cal. 1909, Portland, Victor, Colo. After university course: 1910, Grace Mining Co.'s cyanide plant, West Dip, Mercur, Utah. 1913, Ray Consolidated Copper Co.'s mine, Ray, Ariz.

Present position: Oil flotation process, Ray Consolidated Copper Co.'s mill.

Arthur Dehon Little, Boston, Mass.

Proposed by D. T. Day, A. F. Lucas, E. W. Parker.

Born 1863, Boston, Mass. To 1880, Public schools, Portland, Maine. 1880-81, Berkeley School, New York, N. Y. 1881-84, Mass. Inst. of Tech. 1884-86, Chemist and Supt., Sulphite Pulp Mill, Richmond Paper Co., Providence, R. I. 1886-93, Griffin & Little, Boston. 1893-1900, In general practice, Boston, under own name. 1900-05, Little & Walker, Boston. 1905-09, General practice under own name.

Present position: 1909 to date, Pres. and Genl. Mgr., Arthur D. Little, Inc., Pres. Chemical Products Co., Mansfield Co.

Edward G. Love, New York, N. Y.

Proposed by D. T. Day, A. F. Lucas, E. W. Parker.

Born 1850, New Haven, Conn. 1872, Grad., Academic, Hamilton College. 1876, Grad., Technical, School of Mines. 1878, Ph. D., Columbia Univ. 1878-1913, Analytical and consulting chemist in New York City; Gas Examiner to the City of New York, during a part of that time as chemist to various public and private concerns, State Board of Health, State Dept. of Agriculture, etc.

Present position: Chief Chemist, Consolidated Gas Co. of New York.

Charles H. MacDowell, Chicago, Ill.

Proposed by George S. Rice, H. L. Hollis, F. K. Copeland.

Born 1867, Lewistown, Ill. Common and high school education in home town; home and private instruction scientific subjects, etc. 1883, Commercial course, Wesleyan Univ., Bloomington, Ill. Private study, chemical, engineering, and metallurgical subjects. 1887, Entered service Armour & Co. 1894, Founded commercial fertilizer branch Armour & Co. and subsequently developed various by-product departments. Developed phosphate rock mining branches and much research work in chemical and metallurgical lines for firm, and personally engaged in pyrites and potash developments, alunite, etc.

Present position: President, Armour Fertilizer Works; Tenn. Chemical Co., Nashville, Tenn.; Planters Fertilizer & Chemical Co., New Orleans, and of other Armour interests. Mineral Products Co., New York.

Kenneth Gerard Mackenzie, Bayonne, N. J.

Proposed by D. T. Day, A. F. Lucas, E. W. Parker.

Born 1887, New York City. 1904-07, Course in Chemistry, Sheffield Scientific School of Yale Univ. 1907-08, Researches in organic chemistry, grad. School of Yale Univ. 1907, Ph. B. 1909, M. S., Yale Univ. 1907, Asst. Instructor in Chemistry, Sheffield Scientific School, Yale Univ. 1908, Chemist, Conn. Agricultural Experiment Sta. 1908-10, Research Chemist, New York Testing Laboratory (Barber Asphalt Paving Co.), Maurer, N. J. 1910-11, Chief Chemist, Nairn Linoleum Co., Newark, N. J.

Present position: 1911 to date, Consulting Chemist, the Texas Co.

Robert W. Mackey, Llallagua, Bolivia.

Proposed by Charles A. Chase, Durward Copeland, H. C. Lee, Charles N. Bell, James S. James.

Born 1886, Waukegan, Ill. 1910, Grad. Missouri School of Mines; B. S. 1906-08, Rodman, C. & N. W. Ry., N. W. Elev. R. R., Chicago, Ill. 1910-13, Mucker, Sampler, Asst. Engr., Engr., Liberty Bell Gold Min. Co., Telluride, Colo.

Present position: Chief Engr., Cia. Estanifera de Llallagua.

Theophilus Leeper Nelson, Empire, Ala.

Proposed by H. S. Geismer, Benjamin L. Miller, Henry S. Drinker, H. Eckfeldt, E. A. Smith, J. E. Brantly.

Born 1892, Anniston, Ala. 1911, Grad., Univ. of Ala.; B. S. 1913, One term at Lehigh Univ., graduate work in min. engrg. 1909 and 1910, Summer work as mining engineer's helper, with U. S. Grant, E. M., Empire Coal Co.

Present position: 1911 to date, Mine Supt., Empire Coal Co.

William Frederick Palmer, Jerome, Ariz.

Proposed by B. B. Thayer, J. W. Allen, W. S. Harper.

Born 1888, Manitoba, Canada. 1895-1903, Grammar schools in Butte, Mont.; Washington, D. C.; New York City, and Winthrop, Mass. 1903-07, Boston English High School. 1907-11, Columbia School of Mines, New York; E. M. 1911-12, During winter in Cobalt, Ont., Canada, with Nipissing Mines Co., Ltd., in high grade mill. 1912-13, Asst. Engr., Leonard Copper Co., Gleason, Ariz.

Present position: 1913 to date, Asst. Engr., United Verde Copper Co.

Ray Potter Perry, New York, N. Y.

Proposed by David T. Day, Anthony F. Lucas, E. W. Parker.

Born 1879, Cleveland, Ohio. 1900, Harvard; A. B. 1900-10, Barrett Manufacturing Co., Cleveland, Ohio.

Present position: 1910 to date, Genl. Manufacturing Mgr., general office, Barrett Manufacturing Co., New York, N. Y.

Oluf Gottlieb Petersen, Somerset, Ky.

Proposed by H. D. Easton, J. E. Butler, James Bonnyman.

Born 1876, Erie, Pa. 1897-98, Univ. of Mich., Mech. Engr. 1899-1901, Director Manual Training, Public Schools, Ishpeming, Mich. 1901, Director Manual Training, Milwaukee, Wis. 1902, Supt., Lindenmann & Hover, sheet metal and castings, Milwaukee, Wis. 1902-08, Mech. and Electrical Engr., Stearns Coal & Lumber Co., Ky. and Tenn. Ry., Supt. Motive Power.

Present position: 1909 to date, Pres., Associated Engineering Co.

Roswell Woodworth Prouty, Morenci, Ariz.

Proposed by W. H. Emmons, M. H. McLean, F. H. Hayes.

Born 1890, Grand Forks, N. D. 1904-08, Mechanic Arts High School, St. Paul, Minn. 1908-12, School of Mines, Univ. of Minn., Minneapolis, Minn.; E. M. 1912-14, Min. Engr., Oliver Iron Mining Co., Ely, Minn.

Present position: 1914 to date, Asst. Geol., Detroit Copper Mining Co.

Max F. Quinn, Spokane, Wash.

Proposed by Glenville A. Collins, James F. McCarthy, A. Stanley Hill.

Born 1888, Red Cliff, Colo. 1894-1905, Gonzaga College, Spokane, Wash.; B. A. 1908-12, Univ. of Minn.; E. M. 1906-08, City Engineer's office, Spokane, Wash. 1912, Asst. to Stewart Campbell, Hailey, Idaho. 1913, Idora Hill Mining Co., Wallace, Idaho. 1914, Hecla Mining Co., Wallace, Idaho.

Present position: Engineer and Assayer, Iron Mountain Tunnel Co., Superior, Mont.

Bill Read, San Francisco, Cal.

Proposed by H. L. Haehl, C. E. Gilman, D. M. Folsom.

Born 1884, Sporting Hill, Pa. 1909, Grad., Stanford Univ., Engrg. Dept.; A. B. 1909, California Débris Commission as Surveyor; and Neocene Mining Co. on examination work and sampling. 1910, Tom Reed Gold Mines Co. 1910, Orsk Gold Fields, Ltd., Siberia, as surveyor, sampler, etc. 1912, Surveyor, Haehl & Gilman. 1913, Mt. Whitney Power & Electric Co. 1913, "Forminière," Bruxelles, Belgium.

Present position: Member of firm of Halcombe, Flanders & Read.

George David Rosengarten, Philadelphia, Pa.

Proposed by David T. Day, Anthony F. Lucas, E. W. Parker.

Born 1889, Philadelphia, Pa. 1890, Univ. of Pennsylvania; B. S. 1892, Jena, Germany; Ph. D. 1893-1905, Rosengarten & Sons.

Present position: 1905 to date, Vice-Pres., Powers-Weightman Rosengarten Co.

Herbert A. Sayre, Des Moines, Iowa.

Proposed by P. B. Cronin, S. W. Beyer, McHenry Mosier.

Born 1881, York, Neb. 1900, Grad., West Des Moines High School, scientific and business courses. Summer work at Drake Univ., Univ. of Mo., Univ. of Wis. 1907, B. M. E., and 1912, M. E., Iowa State College. 1901-02, Instructor Manual Training, West High School, Des Moines. 1904, Instructor, North High School, Des Moines. 1907-11, Instructor, West High School, Des Moines. Attended college intermediate years.

Present position: 1911 to date, Treasurer, Eagle Coal & Mining Co.; Supt., Eagle Mine No. 3.

Maximilian Toch, New York, N. Y.

Proposed by David T. Day, Anthony F. Lucas, E. W. Parker.

Born 1864, New York. 1884-86, Grad. and Post-Grad., New York Univ. 1888, Post-graduate courses Columbia Univ., College of Pharmacy. Other special post-graduate courses. LL. B., F. C. S. Author of number of books and original articles on paint, hydrocarbon oils, pigments, cement and concrete, corrosion, etc. For 25 years member and Chief Chemist of the firm of Toch Brothers.

Present position: Chief owner and director, Durex Chemical Works, Sweetwater, Tenn., and owner of about three hundred acres of mine land and mines where barite is mined and converted into various barium salts. Considerable experience in the mining and manufacturing of barium products.

Robert Hinckley Townsend, San Dimas, Durango, Mexico.

Proposed by Philip N. Moore, J. F. Kemp, B. B. Thayer.

Born 1882, Cambridge, Mass. 1895-98, West Roxbury High School, Jamaica Plain, Mass. 1898-1905, Globe Natl. Bk. and Natl. Bk. of Republic, Boston, Mass. 1905-08, Harvard Univ.; A. B. 1908-10, Columbia School of Mines; E. M. A.

Present position: 1910 to date, Mgr., San Luis Mining Co.

Wade Lyndon Wetmore, Anyox, B. C., Canada.

Proposed by A. J. Bone, J. T. Dillon, W. B. Bishop.

Born 1880, Boston, Mass. 1902, Grad., Mass. Inst. of Technology; B. Sc. 1902-03, Hill Clutch Co., Cleveland, Ohio. 1904-05, U. S. Govt., Ordnance Dept., Rock Island, Ill. 1905-06, Allis-Chalmers Co., Mining Dept., Milwaukee, Wis. 1906, Boston Consolidated Mining Co., Salt Lake City, Utah. 1906-07, Balaklava Cons. Copper Co., Coram, Cal. 1907, Utah Cons. Mining Co., Salt Lake City, Anaconda, Mont. 1908-09, Midas Gold Mining Co., Knob, Shasta County, Cal. 1909-12, United States Smelting, Refining & Mining Co., Salt Lake City, Utah, and Kennett, Cal.

Present position: 1912 to date, Engr., Anyox Smelter, Granby Cons. Mining, Smelting & Power Co., Ltd.

Julius I. Wile, New York, N. Y.

Proposed by J. F. Orr, P. A. Mosman, A. Newstaedter.

Born 1877, Rochester, N. Y. 1893, Rochester High School. 1897, Mechanical Engr., Cornell Univ.; Member, Amer. Soc. of Mech. Engrs.; Naval and Military Order; Cornell University Club. 1897, Mechanic, Cleveland Shipbuilding Co. 1898-99, Asst. Engr., U. S. Navy, Rank Ensign, U. S. S. "Boston," Asiatic Station. 1899-1901, Engrg. Dept. Fraser & Chalmers, Chicago. 1901-04, Fraser & Chalmers, London, England. In charge engineering sales dept. 1904-11, Proprietor, Wile Power Gas Co., Rochester, N. Y., and Cleveland, Ohio, mfrs. gas producers for power and fuel. 1911-14, Sales Engr., Chalmers & Williams, Chicago Hts., Ill.

Present position: Designing, equipping and contracting for complete ore-milling plants.

Henry Gordon Williams, Salt Lake City, Utah.

Proposed by Walter Fitch, R. W. Whitley, R. C. Gemmell, H. M. Chance.

Born 1856, Merton, Wis. 1872-75, Wayland Institute, Beaver Dam, Wis., preparatory school for the University of Chicago. 1875-78, Univ. of Chicago, Chicago, Ill. 1880-82, R. R. engineering, Atchison, Topeka & Santa Fe R. R. Co. 1882-85, Supt., Capitol Iron Wks., Topeka, Kan. 1885-91, Practicing mining engineering, coal mines and smelters, Kansas, Colorado, New Mexico, and Utah. 1891-96, Chief Engr. and Asst. Genl. Mgr., Pueblo Smelting & Refining Co., Pueblo, Colo. 1896-1902, Supt. and Genl. Supt., Pleasant Valley Coal Co. (now Utah Fuel Co.), Utah.

Present position: 1902 to date, Genl. Mgr., Utah Fuel Co.

Associate Member

Charles Henry Matthews, Duluth, Minn.

Proposed by C. W. Macdougall, R. R. Seeber, James D. Ireland.

Born 1883, Buffalo, N. Y. 1902, Grad. High School. 1903-06, Ohio Univ., Athens, Ohio. 1906-07, Jeffrey Mfg. Co., Columbus, Ohio. 1907-08, Scioto Valley Traction Co., Columbus, Ohio. 1908-10, Jeffrey Mfg. Co., Columbus, Ohio. 1910-14, General Electric Co., Schenectady, N. Y., Testing, Mining Engrg. and Commercial Departments.

Present position: Commercial Dept., General Electric Co.

Junior Members

Arthur Potter Allen, Flint, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1892, Flint, Mich. 1911, Grad., Flint High School, Flint, Mich. 1911-12, Cost accountant, Buick Motor Co., Flint, Mich.

Present position: Student, Michigan College of Mines.

Thomas Erskine Bassett, Palo Alto, Cal.

Proposed by D. M. Folsom, C. F. Tolman, Jr., J. C. Branner, William H. Shockley.

Born 1890, St. Louis, Mo.

Present position: Student at Stanford University.

George Edward Silver Bayless, Baltimore, Md.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1891, Darlington, Md. Baltimore Polytechnic Institute. Boy's Latin School, Baltimore. Hobart College, Geneva, N. Y. Univ. of Cal., Berkeley, Cal. Charles F. King & Co., Philadelphia. Alaska-Gastineau Mining Co., Juneau, Alaska, mucking to surveying engineering department.

Present position: Student, Michigan College of Mines.

Arthur Eugene Carlson, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1890, Ashland, Wis. 1905-09, High School. 1909-11, Asst. in Laboratory, Lake Superior Iron & Chemical Co., Ashland, Wis.

Present position: 1911 to date, Student, Michigan College of Mines.

Charles Faben Cramer, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1892, Toledo, Ohio. 1913, Toledo High School. 1913, Repairman, Ottawa County Telephone Co.

Present position: Student, Michigan College of Mines.

Joseph David, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1891, Muskegon, Mich. 1905-10, High School. 1913-14, Nevada Consolidated Copper Co. 1913-14, Giroux Consolidated Mines Co.

Present position: Student, Michigan College of Mines.

Morris Claire Drake, Marquette, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1887, Ann Arbor, Mich. 1905, Marquette High School. 1905-08, Accountant Auditor's Office, Duluth, South Shore & Atlantic Ry. 1908-12, Accountant and Cashier, Mary Charlotte Mining Co.; Breitung Hematite Min. Co., Ltd.; Washington Iron Co.

Present position: Student, Michigan College of Mines.

Donald Dickinson Fraser, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1891, Johnstown, N. Y. 1908-12, Johnstown High School. 1913, One year, Union College, Schenectady, N. Y.

Present position: Student, Michigan College of Mines.

Bernhardt E. Heine, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1895, Mt. Clemens, Mich. 1908-12, High School. 1914, Transitman, City of Hancock.

Present position: Student, Michigan College of Mines.

Frank Vyvyan Hicks, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1889, Edwardsburg, Mich. 1908, Grad., Edwardsburg High School, Mich. 1911-12, Ferris Institute, Big Rapids, Mich. 1909, Bitter Root Valley Irrigation Co., Missoula, Mont. 1910, Seattle Electric Co., Seattle, Wash. 1911, With a Deputy Mineral Surveyor, Seward, Alaska. 1912, underground work, Calumet & Hecla Mining Co., Calumet, Mich.

Present position: Student, Michigan College of Mines.

Rowland Bradbury King, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1888, Helena, Mont. Grammar school. 1902, Apprenticeship with C. M. Fassett, Spokane, Wash., assay laboratory. 1907, Asst. Chemist, British Columbia Copper Co., Greenwood, B. C. 1908, Sampler, Takilma Smelting Co., Takilma, Ore. 1909, Assayer, Nevada Treasure Min. Co., Beowawe, Nev. 1910, Chief Assayer and Chemist, British Columbia Copper Co.

Present position: Student, Michigan College of Mines.

Clyde Wallace Nicolson, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1893, Pueblo, Colo. 1905-09, High School. 1909-13, Univ. of Michigan; A. B.

Present position: Student, Michigan College of Mines.

S. H. Zimmerman, Houghton, Mich.

Proposed by F. W. McNair, W. E. Hopper, F. W. Sperr.

Born 1889, Malin, Mich. 1908-10, Ann Arbor High School. 1910-12, Michigan College of Mines. 1912, Timberman, machine man, sampler, shaftman, Veteran Mine, Kimberley, Nev., Nev. Con. Cop. Co., Ruth, Nev. 1913, Miner, Willow Creek Min. & Development Co., Willow Creek, Nev. 1914, Asst. Mech. Engr., Diamond Crystal Salt Co., St. Clair, Mich.

Present position: Student, Michigan College of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Nov. 10 to Dec. 10, 1914. This list, together with the list published in *Bulletin* Nos. 88 to 96, April to Dec., 1914, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of Dec. 10, 1914.

BATTIN, WILLIAM FREDERICK	42 Broadway, New York, N. Y.
BRADFORD, S. K.	527 Waverly St., Palo Alto, Cal.
BRINDLE, A. ST. C.	2189 Saratoga Ave., Victoria, B. C., Canada.
BROWN, CHARLES H.	Oxy-Acetylene & Thermit Welding Co., 212 San Francisco St, El Paso, Texas.
BROWNE, K. C.	Copper Cliff, Ont., Canada.
BURGER, CLARENCE C.	700 Fourth Ave., Asbury Park, N. J.
CARSON, ELLARD W.	Genl. Mgr., Cambria Quicksilver Co., Cambria, San Luis Obispo Co., Cal.
CARTER, THOMAS E.	Box 121, Station A., Chattanooga, Tenn.
CHURCH, JOHN L.	Supt., San Toy Mining Co., Apartado 125, Chihuahua, Mexico.
CLARK, J. M.	509 Brooks St., Charleston, W. Va.
COLE, ROBERT J.	4011 12th Ave., N. E., Seattle, Wash.
COLES, H. O.	372 N. Second Ave., Phoenix, Ariz.
COX, THOMAS	463 Ellita Ave., Oakland, Cal.
CRAMPTON, THEODORE H. M.	607 East Orange Ave., Monrovia, Cal.
CREMER, J. H.	2800 Washington Ave., N. W., Cleveland, Ohio.

- CROWFOOT, ARTHUR.....Care Arizona Copper Co., Ltd., Morenci, Ariz.
 DICKEL, A. C.....P. O. Box B, Goldfield, Nev.
 ECCLES, S. M.....Jamison Coal & Coke Co., Fairmont, W. Va.
 FARISH, FREDERICK G.....315 Colorado Bldg., Denver, Colo.
 FOWLES, CHARLES.....San Lucas de Ocampo, Durango, Mexico.
 GAMBRILL, GEORGE T., JR., Asst. Supt., Barrett Manufacturing Co., Fairfield, Ala.
 GLEESON, WALTER G.....Gleeson Development Co., Granite, Ore.
 GOODLOE, MEADE.....Room 831, 406 S. Main St., Los Angeles, Cal.
 GORMAN, THOMAS C.....7 Delaware Ave., Ottawa, Canada.
 GROSS, JOHN.....541 Corona St., Denver, Colo.
 HARRIS, H., Mgr., Messrs. H. J. Enthoven & Sons, Ltd.,
 Upper Ordnance Wharf, Rotherhithe, London, S. E., England.
 HENRY, W. E., Northwestern Representative, Ore Dept., Empire Zinc Co.,
 Spokane, Wash.
 HOFER, CHARLES A., Villa Bellevue, rue de Bellevue, Suresnes (Seine), France.
 HOFFMANN, ROSS B.....520 Palisade Ave., Yonkers, N. Y.
 HOGARTY, BARRY.....Care American Smelting & Refining Co., Leadville, Colo.
 HUANG, S. KEN, Pinghsiang Colliery, Pinghsiang, via Changsha, Hunan, China.
 HUNT, FRED F.....10-12 Old Slip, New York, N. Y.
 HYDER, CHARLES A.....P. O. Box 395, Cañon City, Colo.
 JOHNSON, CLIFTON B.....Instructed to hold all mail.
 JONES, ZECHARIAH.....Route 1, Box 51, Centralia, Wash.
 JUDGE, ARTHUR T., Mgr., Fairview & C. L. Gold Mining Co., Sheba, Transvaal,
 So. Africa.
 KELLEY, A. L.....Marial, Ore.
 KINNEY, H. D.....Easton, Pa.
 KINZIE, ROBERT A., Room 1009, First National Bank Bldg., San Francisco, Cal.
 LAMB, ROBERT B.....501-502 Traders Bank Bldg., Toronto, Ont., Canada.
 LEE, E. C.....706 N. 12th St., Herrin, Ill.
 LEWISOHN, J. A.....Hotel Utah, Salt Lake City, Utah.
 LONGAN, WALKER B., New York & Honduras Rosario Min. Co.,
 San Juanico, Honduras.
 LOUX, CARL H.....153 S. Johnson St., Pocatello, Idaho.
 MATLACK, E. V.....Westminster Apts., St. Louis, Mo.
 MARR, ROBERT A.....Platinum Mining & Milling Co., Holmes, Wyo.
 MAYNARD, T. P.....Box 121, Station A, Chattanooga, Tenn.
 MERRILL, F. J. H.....631 Higgins Bldg., Los Angeles, Cal.
 MILLARD, H. ALFRED, Cons. Engr., Breitung & Co., Ltd., 11 Pine St., New York, N. Y.
 MILLS, F. P.....Calxico, Cal.
 MOFFETT, CHARLES A.....1500 16th Ave. South, Birmingham, Ala.
 MORRIS, H. C.....The Wyoming, Columbia Road, Washington, D. C.
 MORRISON, W. L.....615 College St., Canonsburg, Pa.
 NEBEKER, A. C.....828 Browning Ave., Salt Lake City, Utah.
 NICHOLS, J. CLAYTON.....Iloilo, Philippine Islands.
 ORR, CHARLES T.....Pres. and Genl. Mgr., Bertha A. Mining Co., Webb City, Mo.
 PATTERSON, R. C., JR.....Kinney Building, Newark, N. J.
 PORTER, JESSE C.....Minas de Matahambre, Cabezas, Pinar del Rio, Cuba.
 PRINCE, ERNEST, Care Westinghouse Elec. & Mfg. Co., Bank of Commerce Bldg.,
 St. Louis, Mo.
 REECE, F. B....."Highfield," Liscard, Liverpool, England.
 REED, DAVID C.....Avenida Bolognesi 271, Barranco, Lima, Peru.
 RHODES, FRED N., Asst. Wks. Mgr., N. Z. Portland Cement Co., Limestone,
 New Zealand.
 RHODES, W. B.....Care Oneida Stag Min. & Mill. Co., Idaho Springs, Colo.
 RICE, GEORGE S.....50 Court St., Brooklyn, N. Y.
 RYDER, ALFRED HAROLD.....55 Kimberley Drive, Crosby, Liverpool, England.
 SANBORN, J. F.....Care Board of Water Supply, 250 West 54th St., New York, N. Y.
 SCHLERETH, C. Q.....911 Mariposa St., Denver, Colo.
 SHALER, M. K.....4 Bishopsgate, London, E. C., England.
 SHUMWAY, RALPH W.....Box 1581, Denver, Colo.
 SMITH, H. I.....Bureau of Mines, Urbana, Ill.
 SPROAT, A. D., Care Cia. de Santa Gertrudis S. A. (Molino Nuevo),
 Pachuca, Hgo., Mexico.
 STONESTREET, GEORGE D.....Instructed to hold all mail.
 STREET, G. H.....446 duPont Bldg., Wilmington, Del.

THROPP, J. E., JR.	243 West School Lane, Germantown, Philadelphia, Pa.
TIERNEY, J. J.	Newtown, Bucks Co., Pa.
TRUSCHKOFF, NICOLAS, Mgr.,	Ekibastous Mines and Smelter,
	Kirgis Mining Co., Ltd., Paslodar, Semipalatinsk District, Russia.
VALENTINE, MALVERN R.	Victor, Colo.
VAN SICLEN, MATTHEW	117 N. Pennsylvania Ave., Webb City, Mo.
WAGNER, WILLIAM HUFF	Milford Del.
WEIGALL, ARTHUR R.	Caxton House, Westminster, London, S. W., England.
WELLMAN, S. T.	1900 E. 90th Street, Cleveland, Ohio.
WELSH, H. LEE	Supt., Oro Fino Co., Southern Cross, Mont.
WIGTON, G. H.	Care Washoe Smelter, Anaconda, Mont.
WILLIAMS, HENRY J.	30 Norfolk Rd., Chestnut Hill, Mass.
WOLF, ALBERT G.	Buckhorn, Nev.
WRIGHT, LOUIS A.	286 Grand St., Newburgh, N. Y.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
AMBROSIUS, J. R.	Guanajuato, Mexico.
ARMAS, MILTIADES TH.	Paris, France.
BODY, J. FRANCIS	New York, N. Y.
CORNELISSEN, JOHANNES	New York, N. Y.
COSBY, F. NEWTON	Pioche, Nev.
CRANDALL, RODERIC	Fazendas Nacionais do Rio Branco, Amazonas, Brazil.
GOEDICKE, CARL	Box 535, San Antonio, Texas.
GOODLAND, GILMORE	17 Gracechurch St., London, E. C., England.
HALL, S. W.	Fredericktown, Mo.
KERR, DAVID GILLESPIE	Toronto, Ont., Canada.
KITSON, H. W.	Care British Columbia Copper Co., Voight's Camp, British Columbia.
MENTZEL, CHARLES	New York, N. Y.
NEWMAN, BRUNO	Apartado 90, Aguascalientes, Mexico.
NORRIE, WILLIAM G.	Three Forks, B. C., Canada.
PALMER, CORTLANDT E.	New York, N. Y.
PINKHAM, W. F.	Battle Mountain, Nev.
RAMBO, WILLIAM C. J.	Denver, Colo.
RANDOLPH, B. S.	Tonoloway Quarries, Tonoloway, via Hancock, Md.
SHEAFE, HARRY J.	El Dorado, Cal.
THOMAS, CHARLES S., JR.	St. Louis, Mo.
WENTWORTH, IRVING H.	245 Belden Ave., Harlandale Addition, San Antonio, Texas.
WHITE, R. T.	Dzansoul, Russia.
WRIGHT, PERCY E.	Seattle, Wash.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Nov. 10 to Dec. 10, 1914.

Date of Election	Name	Date of Decease
1892	*Bolles, J. H.	
1899	**Christian, August	November 19, 1914
1883	*Christy, Samuel B.	November 30, 1914
1906	*Ellam, Albert S.	
1905	*Whipple, J. R.	November 7, 1914
1881	**Wilson, Newton R.	June 23, 1914

* Member

**Life Member.

BIOGRAPHICAL NOTICES

James Ray Whipple was born Oct. 19, 1873, at Decota, Alameda County, Cal., and received his education in the public schools and at the University of California, which he entered in 1896. During his college course he was distinguished in the athletic field, being captain of the University football team in 1900. But this did not prevent him from

graduating with high honors in 1903. His college-mate and athletic comrade, B. L. Thane, whose sister he married while still at the University, subsequently became superintendent of the Eagle River mine in southeastern Alaska; and upon his graduation he went there with his young wife, and became the assistant of Mr. Thane not only in that enterprise but in the superintendency of other mines operated by Eastern companies, such as the Alaska-Gastineau and Kensington. For more than ten years the two men held their outpost of mining industry, winning the admiration, respect and gratitude of their fellow-citizens and employees. In July, 1914, Mr. Whipple found himself the victim of a malignant internal disease requiring surgical relief. But he had just taken charge of the operations of the Alaska-Gastineau Co. at Sheep Creek; and it was nearly two months before he could be persuaded to go to the States for treatment. The rest of his life was a heroic but futile struggle against death, which came at last on Nov. 7. He died and was buried in the little town where he had been born 41 years before. At the time of his funeral on Nov. 10, every man and machine of the Alaska-Gastineau Co. stopped for an hour, symbolizing silently the affection in which he was universally held, and the sorrow with which he was mourned.

Mr. Whipple had been a member of the Institute since 1905.

Newton Richards Wilson, the son of John C. Wilson and Joanna Allen Wilson, was born at St. Louis, Mo., in 1858. While he was yet very young, his parents moved to Brooklyn, N. Y., where he received his early education. Later his family returned to St. Louis, where he completed his education at the Washington University, receiving his degree in 1879. Immediately after graduation, he went to Denver, and during the next few years was actively engaged as a mining engineer, in Colorado and New Mexico, holding several responsible positions. In 1889 he went to Monterey, Mexico, where he built the well known Smelter No. 2, which is still in active operation. This plant he managed for a number of years. In 1900 he went to Torreon, Mexico, where he erected the mammoth Torreon smelter. He was Manager of this plant for some time. Ernest Madero, the uncle of President Madero, was President of the company.

In 1902 Mr. Wilson returned to Monterey to engage actively in the lumber business, in which he had been interested for some years. In 1907 he removed to Beaumont, Texas, where he became Manager of the Industrial Lumber Co. and later its President, which title he held at the time of his death. It is largely due to his efforts and good business judgment that this company has become one of the largest manufacturing concerns of the entire South. At the time of his death, he was also President of the Producers' Turpentine Co. and the Naval Stores Marketing Co., and Vice-President of the Merryville Naval Stores Co.

Having enjoyed almost perfect health all his life, he became suddenly very ill on May 30, and died in St. Louis on June 23, 1914, after an operation for intestinal stricture.

Mr. Wilson was an unusually retiring and reticent man, but possessed the respect and esteem of all who knew him, whether as friends or as business associates. He married in 1897 Miss Sarah L. Glasgow of St. Louis, Mo., who survives him.

He became a member of the American Institute of Mining Engineers in 1881.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Limits of Mining Under Heavy Wash

BY DOUGLAS BUNTING, WILKES-BARRE, PA.

(New York Meeting, February, 1915)

THE first presentation of this paper was before the Pennsylvania Anthracite Section of the Institute in May, 1914, after which a committee was selected to verify and add to the data contained in the original paper. The author is indebted to G. W. Engel, H. W. Montz, R. V. Norris, and H. H. Otto, of this committee, for additional information. I also wish to acknowledge the courtesies extended me by a number of the mining companies for information relative to accidents, systems of proving, and mining practice. The paper should bring forth additional criticism and recommendations, which will be of value in establishing future practice relative to this subject.

Throughout the northern anthracite field of Pennsylvania glacial and alluvial deposits of sand, gravel, and clay exist in varying quantities. The extent and character of these deposits are described by N. H. Darton in *Bulletin* 45 of the U. S. Bureau of Mines, entitled, Sand Available for Filling Mine Workings in the Northern Anthracite Basin. It is estimated that the northern field contains 10,000,000,000 tons of these deposits, or a sufficient quantity to cover the 176 square miles of coal area to a depth of about 25 ft. These deposits of sand, etc., overlying the coal measures attain a maximum depth of more than 300 ft., and are usually saturated to such an extent as to render them semi-fluid. It is this condition of fluidity that limits the mining of coal seams cropping in or in close proximity to these deposits, and it is in the interest of safer and more efficient mining methods under these conditions that the study of this subject is here presented.

The dangers incident to mining under deep deposits of sand have long been realized by operators in the northern anthracite field, and much has been done to prove the contour of the rock and vein croppings underlying these water-bearing deposits of sand and gravel.

Although, as far as we know, the accidents of the past have not resulted in the loss of life, except in two instances, yet nearly all were attended with more or less miraculous escapes of men working in the vicinity of the breaks, so that the probable existence of pot holes, etc.,

brings to our attention most forcibly the necessity for exercising every care and precaution in working mines in territory of this character.

TABLE I.—*Accidents in Wyoming Field Due to Inrushes of Sand and Water*

Accident, No.	Date	Mine	Location	Vein	Tapped by	Result
1	July 4, 1872	Burroughs	Plains- ville	Hillman	Breast	An inrush of sand and water. A pumpman, the only man in the mine at the time, easily escaped.
2	June 30, 1874	Wanamie No. 18	Wanamie	Red Ash	Breast	Gangway and workings in the vicinity were filled for some distance from the break.
3	Jan., 1882	Maltby	Swoyers- ville		Rock Plane	Gangways were filled; also the shaft for a vertical height of 90 ft.
4	Apr. 23, 1884	Fuller	Swoyers- ville	Six Foot	Slope	Slope filled to the top for a distance of 900 ft.
5	 1884	Ridge	Archbald	Archbald		
6	May, 1885	Ridge	Archbald	Archbald		
7	Dec. 18, 1885	No. 1 Slope	Nanti- cooke	Ross	Breast	Gangways in the vicinity were completely filled in less than an hour.
8	Aug., 1889	Fuller	Swoyers- ville		Rock Plane	The plane and all workings tributary to it were filled with sand or water.
9	Mar. 1, 1897	Mt. Look- out	Wyoming	Pittston	Breast	A large area of the workings was filled; no men at work at the time.
10	Dec. 30, 1898	Wanamie No. 18	Wanamie	Cooper	Breast	Gangway on lower level filled to a height of 2 ft. for a distance of 300 ft. Depression on the surface 100 ft. east and west and 75 ft. north and south.
11	Feb. 2, 1899	Franklin	Wilkes- Barre	Kidney	Breast	Filled gangway to a height of 3 or 4 ft. for a long distance.
12	Apr. '13, 1899	No. 2 slope	Nanticoke	Hillman	Breast	Gangways were filled for several thousand feet. Breasts had been worked 26 years previous; no men at work in the vicinity. Surface depression was 70 to 80 ft. deep.
13	Apr. 25, 1899	Bliss	Hanover	Hillman	Breast	Gangways and tunnel in the vicinity were filled tight to roof. Conical depression on surface 60 ft. in diameter and 40 ft. deep.
14	June 10, 1914	Sugar Notch No. 9	Sugar Notch	Kidney	Breast	Gangways and tunnels were filled tight to the roof. Depression on surface 150 ft. wide, 210 ft. long and 60 ft. deep.

To safeguard the mine workings against possible danger, some of the mining companies have formulated rules for minimum thickness

of rock to be maintained under water-bearing wash. These rules are variable. Some companies work to 100 ft. of rock over the vein when the depth of wash exceeds 100 ft., and for less than 100 ft. of wash the thickness of rock is maintained the same as the depth of wash to and including 40 ft.; others work to a uniform thickness of rock, varying from 20 to 100 ft., for all depths of wash. Then again, with the progress of mining and the more definite and closer proving of the rock contour, the minimum rock thicknesses previously established are reduced and the workings carried closer to the overlying sand deposits.

A number of accidents have occurred, due to mine workings striking into deposits of sand, etc. For the purpose of reviewing the results of these inrushes, also to compare the conditions thought to exist at the time with those subsequently proved, Table I is presented, in which the more important accidents, of which any record can be found, are given.

Accident No. 1 occurred in a breast in the Hillman vein. Its exact location does not seem to be known, but is presumably within 200 ft. of the old canal bridge. It has been stated that the cave occurred under the old canal located south of the Burroughs shaft. The canal bed over these old workings, which are also known as the Enterprise workings, ran on a course which would maintain a practically constant depth to the Hillman vein of about 68 ft. There have been no provings of the top of rock along this old canal bed in this vicinity, consequently nothing is known as to the probable thickness of rock over the vein at the point of inrush.

Some time in October, 1882, another cave occurred in the Hillman vein of the Mitchell shaft workings within 1,500 ft. of the cave which occurred on July 4, 1872. The river had risen to a considerable height so that the entire surface of the "flats" in that vicinity was covered with water. It was while this condition existed that the cave occurred. The opening on the surface was about 150 ft. long by 50 ft. wide. On Mar. 29, 1913, when the river had again risen to flood height and covered this same territory, a cave 110 ft. long and 90 ft. wide occurred in the identical spot. Again, on Mar. 30, 1914, the river having flooded this area, another cave occurred in the same location.

The section, Fig. 1, shows the old Hillman workings in the vicinity of this cave. The vein had been worked to a point very near the top of rock and the records of drill-hole provings show the rock to be very soft. It is evident that the 70 ft. of overlying wash was too heavy for the small amount of soft roof rock to withstand.

In Fig. 2 is shown a cave which occurred in 1876 in the Hillman vein, near Port Bowkley, causing a depression on the surface about 100 ft. in diameter. The vein is on a 15° pitch and was worked by chambers to a point afterward discovered to be within 15 ft. of the top of the rock. At this point a break in the rock was encountered which allowed a large

volume of water, together with sand and gravel, to enter the workings. Eventually the roof gave way across the width of the chamber, the opening afterward being found to be about 20 ft. in diameter, and a large quantity of sand, culm, and gravel was deposited in the lower gangways, filling the workings to the shaft level.

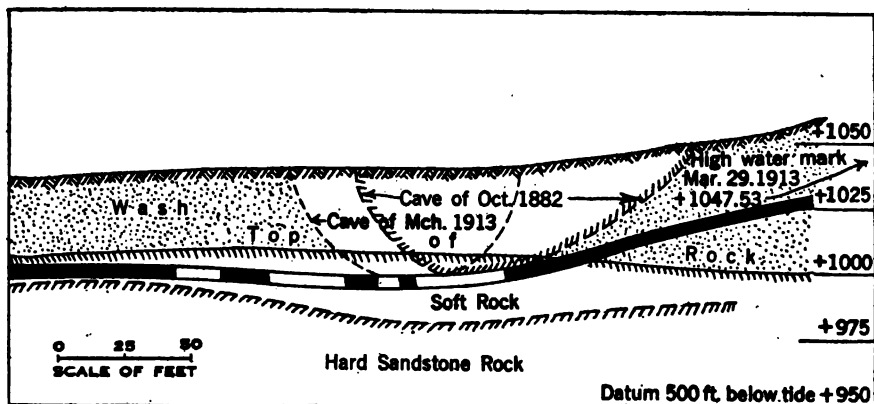


FIG. 1—PROFILE OF HILLMAN VEIN WORKINGS.

In 1890 it was decided to remove the water and pumps were installed to do the work. When practically all the water had been pumped out of the workings, it was decided to build a dam as near as possible

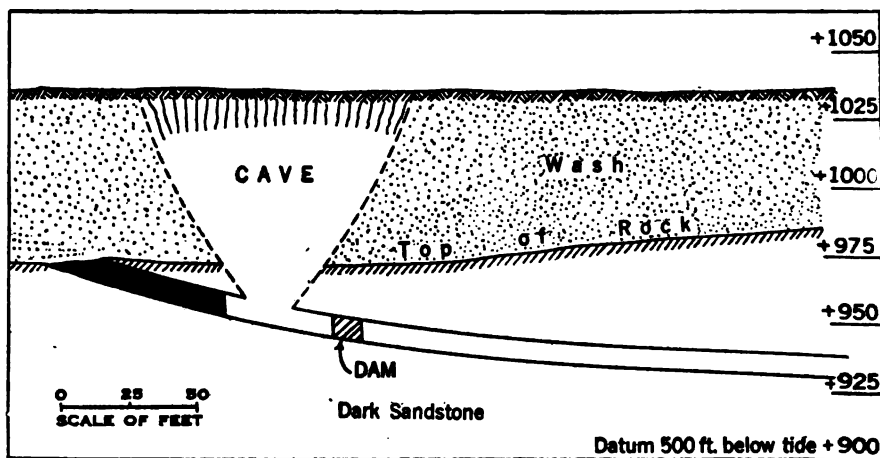


FIG. 2.—CAVE TO HILLMAN VEIN, IN 1876.

to the cave hole. In carrying out this scheme a counter gangway was driven to avoid cleaning up the gangway. It was decided to build a brick wall, as shown in Fig. 3, the ends of which were to be set in comparatively large pillars. In order to accomplish this, a number of small

pillars of coal had to be cut to allow the wall to pass through. The brick wall was 80 ft. long by 13 ft. high by 4 ft. thick. As the overlying strata of the Hillman vein were of a slaty nature, it was decided to blow down the roof in steps in order to afford the wall good support. The same was done with the bottom rock. The wall was arched to the inside from top to bottom. In order to drain the water out of the dam from time to time, it was deemed necessary to place holes in the wall. These holes were 16 in. square on the outside and 24 in. square on the inside, and were closed with wooden plugs, which were operated from the outside of the dam as found necessary.

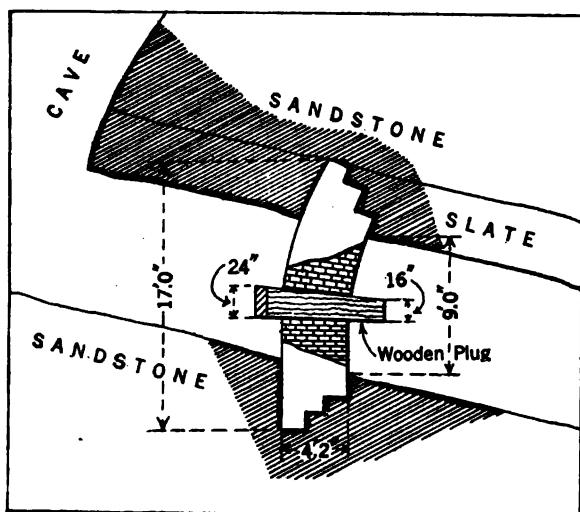


FIG. 3.—DAM IN HILLMAN VEIN, PORT BOWLEY.

Accident No. 2 occurred in the face of a breast in the Red Ash vein. One man was killed some distance out on the gangway by being caught in the rush of sand and water. The breast in which the cave occurred was approaching an anticlinal. The face was approximately 115 ft. below the surface. Subsequent provings of the depth of sand in this vicinity indicate that an eroded channel exists on the line of the anticlinal with a width of approximately 150 ft. and a depth of 40 ft., or a depth below the surface of 65 ft. This would leave from 30 to 35 ft. of rock over the vein at the point of inrush, so it is presumed that a pot hole existed over the face of the breast at that depth.

Accident No. 3 occurred in the face of a rock plane being driven from the Eleven Foot vein to the Six Foot vein, but instead of striking the coal, it broke into sand, which in a few hours filled the mine and shaft to within about 65 ft. of the top. A drill hole 300 ft. from the break indicates approximately 14 ft. of rock over the vein, so the presump-

tion that the vein was eroded at the face of the rock plane is probably true. The depth from the surface to the top of rock is approximately 160 ft. and the depression produced on the surface was 150 ft. in diameter. A number of attempts have been made to seal the rock plane by injecting cement grout through bore holes, but without success. The elevation at which the water stands in the shaft indicates that the head of the water-bearing strata in the cave hole is approximately 65 ft. below the surface at this point. A variation in the elevation of the water in the shaft might naturally be expected to occur with any variation in the head in the water-bearing strata, which in turn could vary with the elevation of the water in the Susquehanna river.

Accident No. 4 occurred in the face of a slope immediately following the firing of a shot. The inrush of sand and water filled the slope to the shaft level, which is practically at the same elevation as the surface directly above the face of the slope. The slope has never been recovered. Drill holes in the vicinity indicate that there may have been 20 ft. of rock over the vein at the point of failure. The vertical distance from the face of the slope to the surface is 100 ft.

Accidents Nos. 5 and 6 were due to the presence of pot holes located about 1,000 ft. apart. In a paper by C. A. Ashburner,¹ pot hole No. 1 is said to be 38 ft. deep, 24 ft. wide by 42 ft. long on the surface and 17 ft. wide by 14 ft. long at the top of the vein. At a distance of $4\frac{1}{2}$ ft. above the vein the width is given as 18 ft. and the length as 19 ft. The hole is in slate and sandy shale and the faces are extremely smooth.

Accident No. 7 occurred near the face of a counter gangway close to an anticlinal, and caused the death of 26 men, whose bodies were never recovered. On the surface above the point of inrush there was a culm bank and the depression due to the cave made a cone-shaped hole on the culm bank about 300 ft. in diameter. It has been proved by drilling close to the point of cave that the top of rock is 261 ft. below the top of the culm bank, 200 ft. being sand and 61 ft. culm. The exact thickness of rock over the vein at the point of failure is not definitely proved, but is probably about 22 ft. and positively not more than 48 ft. To the northwest of the location of this accident a considerable area was worked in the same vein without accident, although there exists but 40 ft. of rock over the vein and 174 ft. of sand and culm.

The culm bank under which this cave occurred is located on top of a hill, surface elevation + 650. Rock outcrops are found both above (+ 643 and + 594) and below (+ 567.5) the bank, and the creek valley, where deep wash might reasonably have been expected, is over 1,000 ft. south of the point of cave. The elevation of the top of the culm bank over the cave was + 697, and of the surface at the creek about + 540; the roof of workings at the point of cave, + 388. Sub-

¹ *Trans.*, xv, 636 (1887).

sequent borings in the creek valley showed a rock elevation of + 442, while the rock at the point of cave was not over + 436 and probably about + 410. Considering the rock exposure, elevation + 567, between these two, it is apparent that the buried valley split, and that besides a deep and unsuspected gorge under the hill there was probably a pot hole at the point of cave.

Accident No. 8 occurred in the face of a rock plane being driven from the foot of a slope in the Eleven Foot vein to the Six Foot vein, and at a point 1,300 ft. south of the location of accident No. 4. The plane had been driven 225 ft. on about 14° when the inrush occurred. There is no record to show that the plane struck into the Six Foot vein, but, considering the elevation to which the plane had been driven, it appears that the vein was eroded at this point, although it was thought that 40 ft. of rock existed over the vein. The depth from the surface to the face of the plane is approximately 150 ft. The inflow of sand and water filled the Eleven Foot vein workings and the shaft up to 10 ft. above the Six Foot vein, which is about the same level to which the inrush of accident No. 4 filled the Six Foot slope. The Six Foot vein in the shaft is about 15 ft. lower than the surface above the face of the rock plane, so it is evident that the water in the shaft was at the same elevation as the water-bearing strata in the vicinity of the cave.

There are reasons now to believe that an abrupt washing out of the rock was the cause of this accident as well as of accident No. 4. Within a very short distance north of these two caves there appears to be sufficient cover. The water in the Fuller mine is being lowered, and it is presumed the openings leading to the water-bearing strata are largely closed, as the water has now been lowered to a point about 16 ft. below the Six Foot landing in the shaft.

Accident No. 9 occurred in the face of a breast approaching an anticlinal, about 5 hr. after the miner had left. The place had become very wet and the water percolating through the roof was evidence of the proximity of a deep erosion, as a bore hole some distance in advance proved that 46 ft. of rock and 86 ft. of sand overlaid the vein. The breast was being driven 24 ft. wide in the Pittston vein, which has a thickness of 6 ft. Eight brick dams have been built to guard against any future starting of the sand. The building of these dams has sealed off about 13 acres of territory.

Accident No. 10 occurred in the face of a breast just started off a counter gangway. Five men were at work close by but all escaped. The vein at this location pitches 50° , and the counter gangway from which the breast was started is 95 ft. below the surface. The presence of a dangerous deposit of sand was never suspected, as the rock croppings on the surface were only 100 ft. south of the line of the gangway. Subsequent provings of the top of rock indicate that the normal depth

of sand is 55 ft., which would leave 40 ft. of rock over the gangway. The break occurred at about 15 ft. above the gangway, so there must have been a pot hole or an eroded gorge on or near the crop of the vein, with a depth of approximately 25 ft.

Accident No. 11 occurred in the face of a breast in the Kidney vein, which at this point pitches about 25°. The face of the breast became very wet and was stopped two or three days before the accident. Subsequent to the inrush a battery was built in the breast a short distance below the face and the depression on the surface filled.

In order to show how serious this accident might have been, on the night of the cave, no less than 100 people had been enjoying the skating on a pond immediately overlying this area. When the cave occurred, large pieces of ice were carried into the workings along with the mud, which filled the gangways to a depth of 3 or 4 ft. Large pieces of ice were discovered later at a distance of about 1,000 ft. from the cave. The cave occurred about 2 A.M. In the chambers east of where the cave occurred the fire clay was taken down and the "bone" propped up to afford better roof. Batteries were built near the faces of the chambers in order to withstand any possible danger of another cave.

A drill hole put down from the surface at a point 80 ft. west of the cave indicates that the maximum thickness of rock over the vein at the face of the breast in which the break occurred was 15 ft. and it is probable that it was considerably less. The depth from the surface to the point of inrush is 67 ft. On Feb. 27, 1908, a second cave occurred at the same point, presumably due to the battery, erected nine years before, giving way. This second cave produced a cone-shaped depression on the surface with a maximum diameter of 140 ft. and a depth of 40 ft.

Accident No. 12 occurred in an old breast which had been driven 26 years earlier. The face of the breast is 127 ft. below the surface and it was thought that the rock overlying the vein had a thickness of about 50 ft., but it has subsequently been proved to be not more than 20 ft. and was probably less.

This accident occurred at a point about 1,200 ft. south from accident No. 7. In Fig. 4 are shown the conditions as developed by subsequent drilling. The cave hole extended over $3\frac{1}{2}$ acres of surface, and the quantity of clay, gravel, and quicksand washed into the mine approximated 400,000 cu. yd., and reached points in the workings over 4,000 ft. from the point of cave. The great amount of débris washed into the mine was largely due to the fact that the cave occurred at about the junction of Forge and Newport creeks, so that the waters of both flowed into the cave and washed in large amounts of material beyond that naturally tributary to the opening. The flow was finally stopped by throwing in large quantities of brush, prop timbers, hay, etc., and checking the flow of the two creeks by temporary dams.

Accident No. 13 occurred immediately following the firing of a blast. Men at work in the vicinity escaped.

Accident No. 14 occurred in the face of a breast being driven in the

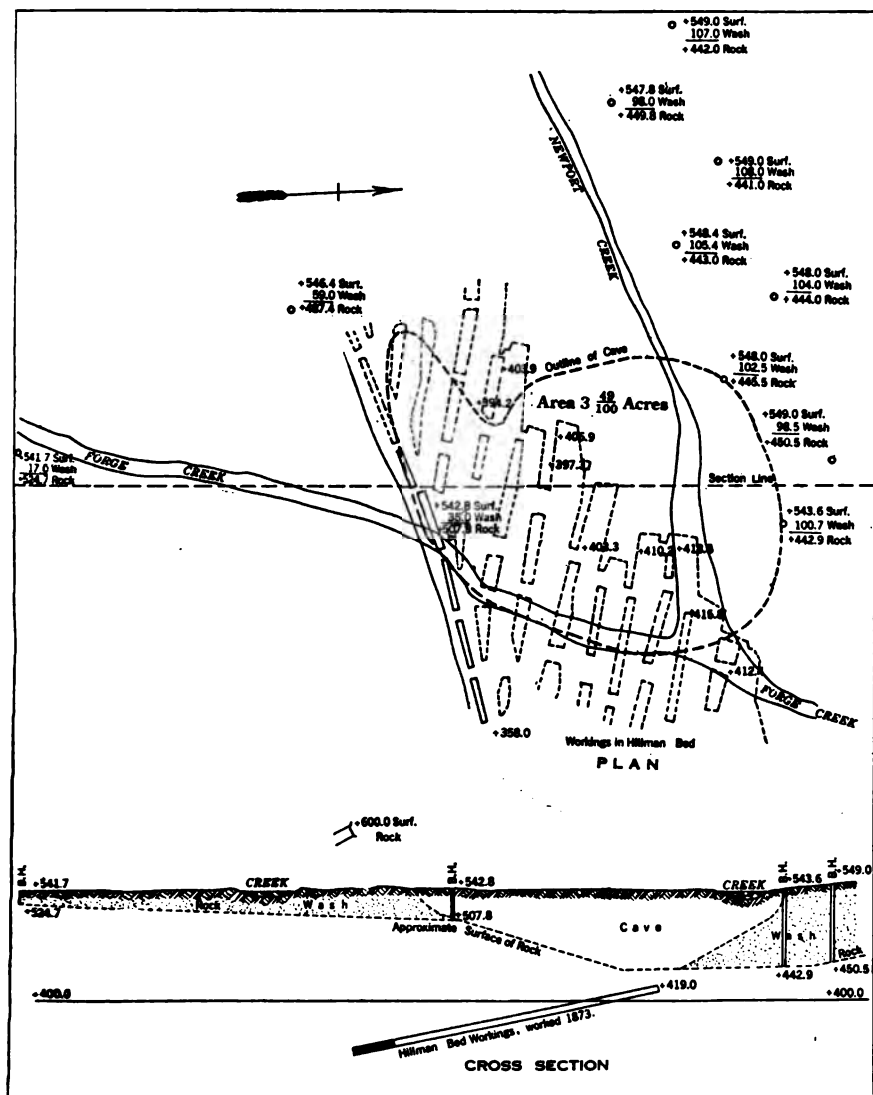


FIG. 4.—PLAN AND CROSS-SECTION SHOWING LOCATION OF ACCIDENT NO. 12,
APRIL 13, 1899.

Kidney vein, immediately following the firing of a blast. Men working in the vicinity were immediately notified; those working on the inside of the cave escaped through a hole in the rock leading from the Kidney

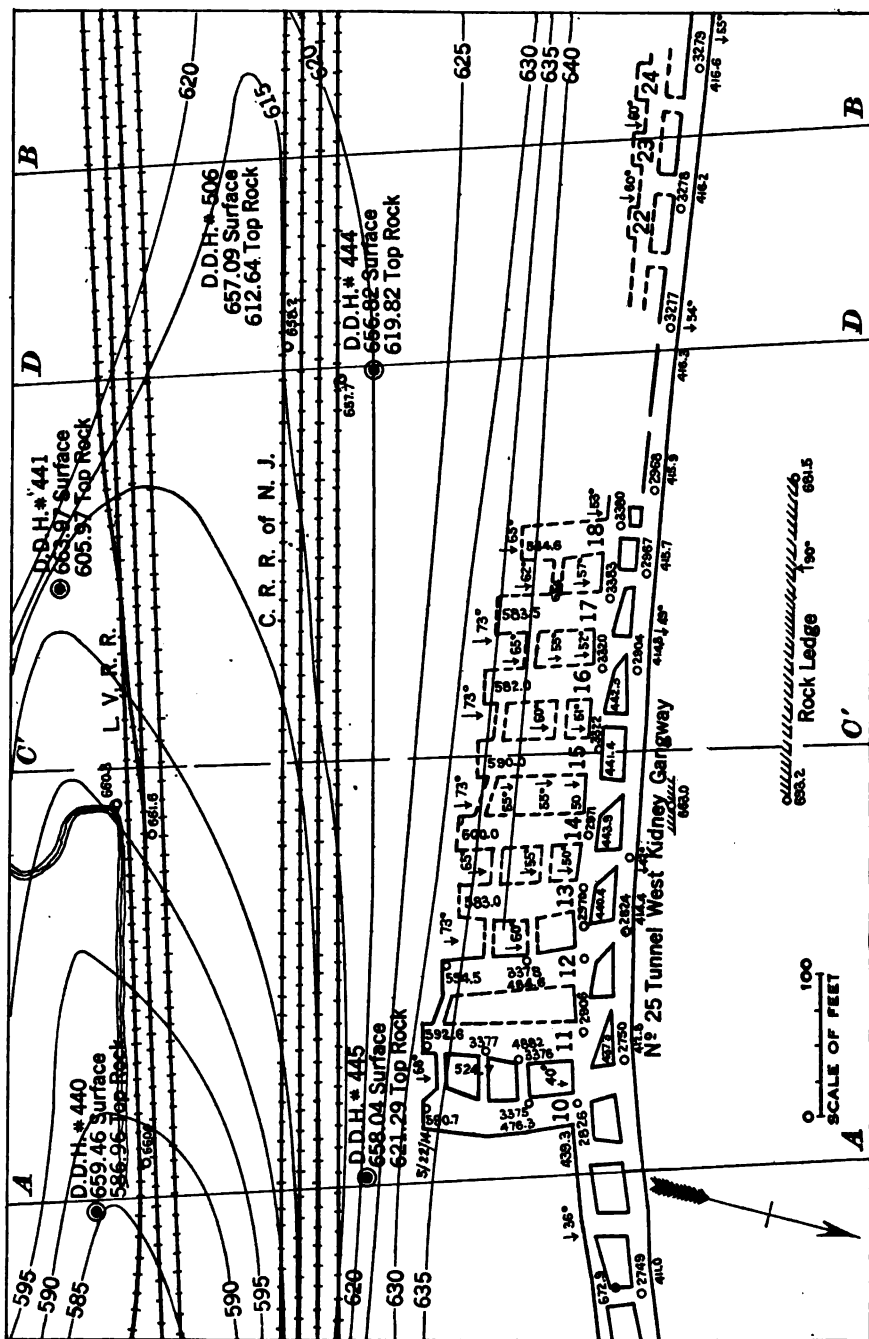


FIG. 5.—MINE WORKINGS AND SURFACE FEATURES PREVIOUS TO ACCIDENT NO. 14.

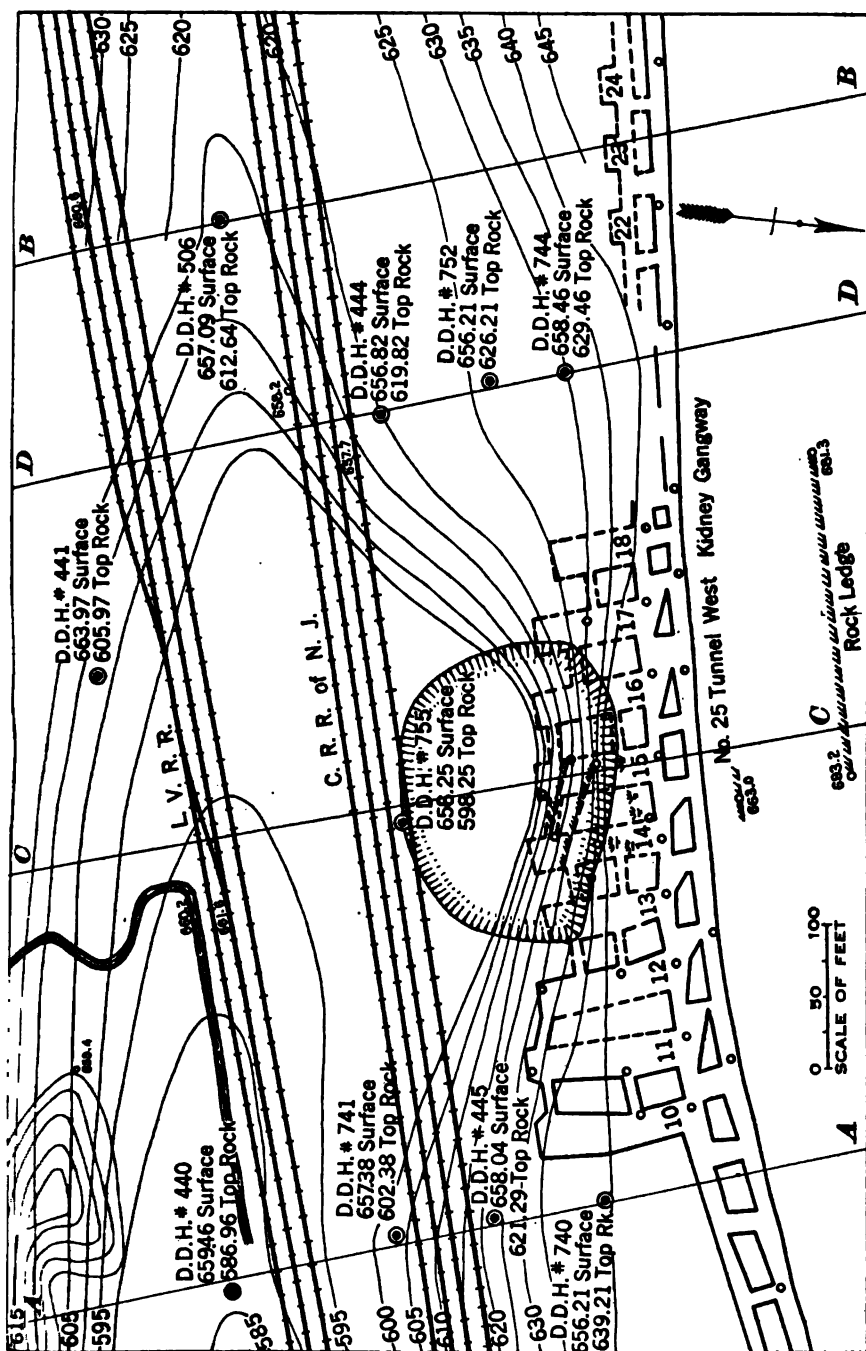


FIG. 6.—SURFACE CONDITIONS SUBSEQUENT TO ACCIDENT NO. 14.

vein to the surface at a considerable distance inside of the cave, and those working on the outside of the cave escaped to the hoisting shaft. The cave occurred about 11 o'clock in the morning, the first evidence on the surface being a conical hole which continued to increase in size for 5 or 6 hr. The material carried into the mine consisted principally of sand and clay in a semi-fluid state, from a swamp in which the cave occurred.

Approximately 20,000 cu. yd. of solid material was carried into the mine, which filled or partly filled several thousand feet of gangways and tunnels and required months to clean up.

In Fig. 5 are shown the mine workings, contours of top of rock, location of drill holes, and the principal surface features as they existed

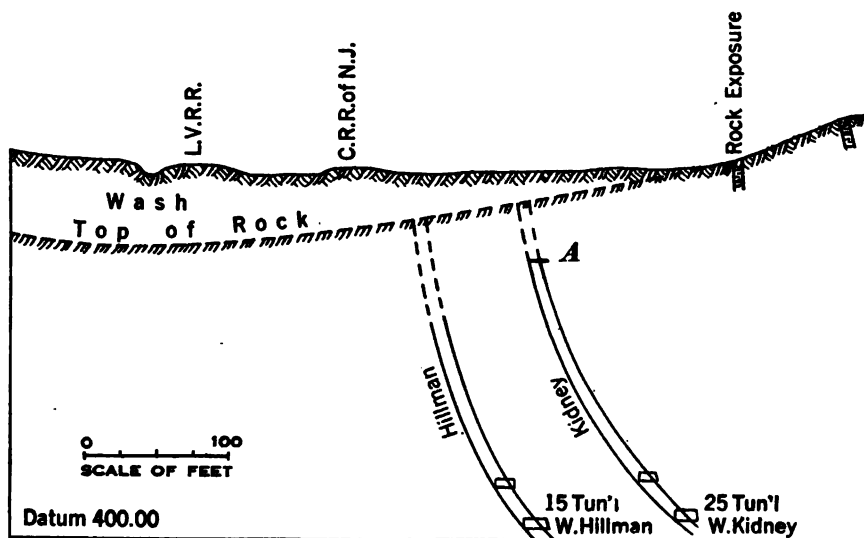


FIG. 7.—SECTION ON LINE C'-C' OF FIG. 5.

previous to the accident. The cave occurred at the face of breast 15, where the vein pitches 75° .

In Fig. 6 are shown the conditions after the cave. It is to be observed that additional drill holes were put down and more accurate contours of the top of rock secured.

In Fig. 5 and Fig. 7, section C'-C' illustrates the conditions as they were considered to exist before the accident. The face of breast 15 at the time of the cave was at A, or at an elevation of + 590.0. It was intended to drive this breast to an elevation of + 600.0, as the elevation of the top of rock immediately over this breast was thought to be + 630.0. The elevation of the surface directly above is + 657.0.

In Fig. 8, section C-C illustrates the conditions after the accident

which conditions are also shown in plan in Fig. 6. The top of rock at the point of cave is at elevation + 600.0, which is only 10 ft. above the face of the breast at the time of the accident.

The difference in the contour of the top of rock on these two sections is very striking, still the top of rock on sections approximately 300 ft. east and west of section C-C has not been materially changed by drill-hole provings made subsequent to the time of the accident.

At the Exeter colliery, near West Pittston, in proving the top of rock in advance of mining in the Pittston vein, a pot or gorge was developed. This erosion has a maximum depth of 110 ft. and a width on top of about 200 ft. The depth from the surface to the bottom of the pot or gorge is 127 ft. Two holes 100 ft. apart and at right angles

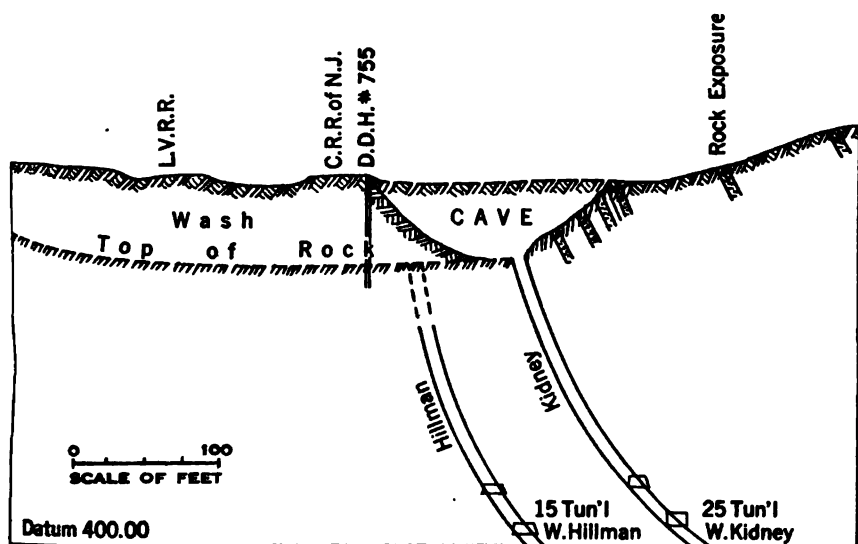


FIG.—8. SECTION ON LINE C-C OF FIG. 6.

to the holes proving a width of 200 ft. show the same depth of erosion, so it appears that it may be an eroded gorge, but if a pot, it is probably elliptical in shape. An indication that the erosion is a gorge is given by considering the rock measures. The top of rock immediately north of the erosion is apparently the bottom rock of the Checker vein and at the location of the deep erosion there is apparently a broken anticlinal, which would give cause for the deeper erosion. The proving of top of rock in this locality was being done with holes spaced 200 ft. centers when the deep erosion was discovered.

A very interesting proving of rock erosion at the Stearns colliery, near Nanticoke, indicates the presence of a deep gorge with a width on top of about 100 ft. and a depth on one side of 95 ft. and on the other

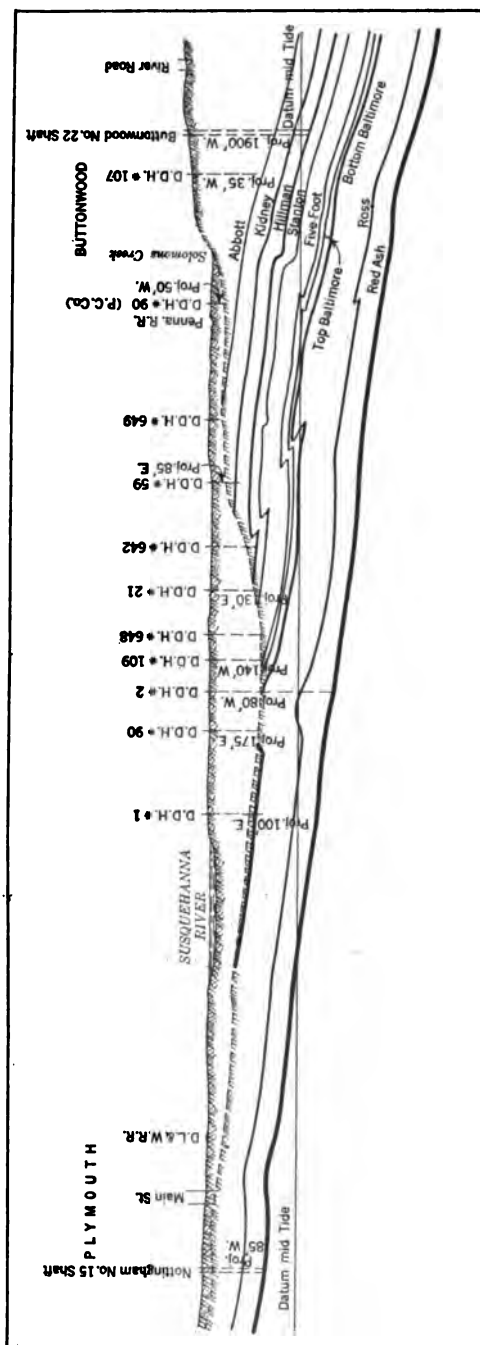


Fig. 9.

side of 45 ft. The bottom of the gorge is 105 ft. below the surface. The gorge as developed indicates, in consideration of provings of the top of rock on sections to the east and west, that it is a contraction in the eroded channel, and like the so-called pot hole over the Pittston vein at the Exeter colliery, is probably nothing more than an eroded gorge. These possible conditions exemplify the importance of contouring the top of rock in provings of rock erosions.

In Fig. 9 is shown a cross section of the Wyoming valley from Plymouth to Buttonwood, on a line west of the Nottingham and Buttonwood shafts. This section is through the deepest point proved in the buried valley, where a depth of 309 ft. of sand, etc., was proved. The territory proved in this part of the buried valley is approximately 2 miles long and 1 mile wide and to date has required the drilling of more than 200 holes. The holes are spaced approximately 500 ft. apart and the various veins approaching the sand deposit have been worked to a uniform rock cover limit of 100 ft.

In proving the top of rock by drilling, some of the mining companies have pursued definite methods of hole spacing. Some companies space the holes on 200-ft. squares, except where the depth of sand is comparatively shallow, when the holes are spaced on 100-ft. squares. Others have spaced the holes approximately 200 ft. apart on the drilling section lines, which section lines are parallel and approximately 500 ft. apart. Additional holes are drilled intermediate to the section lines to prove any probable deeper wash. Other companies have followed no definite rule as to hole spacing. The depth to which holes are drilled into the rock is almost universally 10 ft., except where holes are drilled deeper for the purpose of proving the location and character of some underlying coal vein.

It is advisable to have all measurements of drill holes and the cores checked by a representative of the mining company, also the drilling of at least 5 ft. in the rock should be witnessed by the same representative before finally measuring the hole.

In developing the top of rock under sand deposits by drilling, it is advisable to construct sections on the lines of drilling and also contour maps of the top of rock. Such sections and contour maps are of great value in proving the possible existence of gorges or abrupt erosions, and at the same time are of further value in determining the most desirable locations of drill holes and the lines of working limits for definite rock cover.

Numerous tests of various stones have proved that sandstones take permanent sets for the smallest loads, whereas granite and limestones are nearly perfectly elastic. It has also been proved by tests on various stones that the modulus of elasticity in compression is practically the same as in cross bending, but no fixed relation has been deter-

mined of the compressive, tensile, or shearing strength of the various kinds of stone.

The shearing strength of sandstones and slates per square inch is generally slightly in excess of the modulus of rupture, and the compressive strength of various stones is variable and of comparatively little consequence here, as the compressive strength of even the lightest sandstones ranges from 4,000 to 6,000 lb. per square inch.

The moduli of rupture of various kinds of stone as given by a number of authorities are shown in Table II.

TABLE II.—*Moduli of Rupture of Stones*

	Maximum	Minimum	Average	Authority
Blue stone flagging.....	4,511	360	2,700	Baker.
Slate.....	9,000	1,800	5,400	Baker.
Slate.....	11,230	7,425		Arsenal tests, 1902
Slate.....			8,480	Merriman.
Granite.....	2,700	900	1,800	Baker.
Granite.....			1,754	Merrill.
Granite.....			2,610	Arsenal tests, 1907.
Granite.....			1,667	Arsenal tests, 1905.
Granite.....			1,365	Bauschinger.
Granite.....			1,194	Bauschinger.
Glass.....	3,500			Church.
Glass.....	4,132			Fairbairn.
Sandstone.....			1,576	<i>Technology Quarterly.</i>
Sandstone.....			1,200	Merriman.
Sandstone.....	1,273	655		Arsenal tests, 1895.
Sandstone.....	2,243	1,500		Arsenal tests, 1895.
Sandstone.....	2,340	576	1,260	Baker.
Sandstone (variegated).....			469	Bauschinger.
Sandstone (variegated).....			718	Bauschinger.
Sandstone (variegated).....			1,109	Bauschinger.
Sandstone (variegated).....			341	Bauschinger.
Sandstone (Carboniferous).....			483	Bauschinger.
Sandstone (slaty).....			249	Bauschinger.
Sandstone (slaty).....			135	Bauschinger.
Sandstone (green).....			156	Bauschinger.
Sandstone (Cretaceous).....			597	Bauschinger.
Sandstone (Cretaceous).....			967	Bauschinger.
Gray stone.....	2,200			Kent.
Light stone.....	1,170			Kent.
Stone.....			2,000	Merriman.
Quartz conglomerate.....			654	Merriman.

From the results of tests as given in Table II, the average moduli of rupture for the various stones can be considered as follows:

	Pounds per Square Inch
Blue stone flagging.....	2,700
Slate.....	7,736
Granite.....	1,681
Glass (maximum).....	3,866
Sandstone.....	806

Safe unit stresses for various stones have been given by many authorities and I give in Table III the stresses in pounds per square inch recommended by W. J. Douglas as illustrative of possibly a fair average of such values.

TABLE III.—*Safe Unit Stresses for Stone*

	Compression Pounds per Square Inch	Shear Pounds per Square Inch	Tension Pounds per Square Inch
Blue stone flagging.....	1,500		200
Granite.....	1,200	200	150
Limestone.....	800	150	125
Sandstone.....	700	150	75

It is to be observed, in the case of sandstone, that a safe tensile strength of 75 lb. and a shearing strength of 150 lb. per square inch are given. Now, in consideration of the fact that the modulus of rupture is invariably in excess of the tensile strength, also that the resistance to shear slightly exceeds the modulus of rupture, a value of 100 lb. per square inch for the modulus of rupture of sandstone would be consistent. This assumed value of 100 lb. per square inch as the modulus of rupture for sandstone beams compared with the average modulus as derived from Table II gives a safety factor which, although slightly less than the factors generally used for stone in building construction, seems to be sufficient for the conditions of this problem.

When sandstones and slates, which generally overlie the coal veins, are considered as beams or slabs spanning mine openings for the support of overlying strata or other superimposed load, their transverse strength is of first importance. The ability of such material to serve as a beam depends upon its tensile strength, since that is always less than its compressive strength.

In the formula for flexure $\frac{pI}{e} = M_m$, we can compute the value of p , the greatest normal stress in any outer element, when all other quantities are known. This value of p is theoretically correct provided the elastic limit of the material has not been exceeded. This formula has also been used for the rupture of beams by flexure by calling the value of p thus obtained the modulus of rupture, R . The value of R may be found to differ considerably from the unit tensile strength of some materials and is frequently much larger, having a value between the

ultimate tensile and the ultimate compressive strength. This might be expected, since, even supposing the relative extension of the fibers to be proportional to their distances from the neutral axis as the load increases toward rupture, the corresponding stresses, not being proportional to these strains beyond the elastic limit, no longer vary directly as the distances from the neutral axis; and the neutral axis does not necessarily pass through the center of gravity of the section. It is evident that the modulus of rupture does not express the actual stress in the extreme fiber of the beam, but is a quantity useful only as a basis of comparison.

The manner of failure under test depends upon the kind of material. Stones and brittle materials fail by the fibers breaking in tension, whereas, timber and the more plastic materials fail by crushing. Stone and plain concrete beams of deep section fail sometimes by shearing on a diagonal plane near the point of support.

In deriving a formula for computing the breaking load of a slab of stone from the formula $\frac{pI}{e} = M_m$, let W represent the distributed load including plus the weight of the beam itself in pounds; let b , d , and L represent the breadth, depth, and span, respectively, in inches; let R equal the modulus of rupture, in pounds per square inch.

The maximum bending moment for a constrained or prismatic beam is equal to $\frac{WL}{12}$. By substituting in the formula for flexure $\left(\frac{pI}{e} = M_m\right)$ we obtain the formula $W = \frac{2bd^2}{L} R$. Likewise, the maximum moment at the center of such a beam being equal to $\frac{WL}{24}$, the formula becomes $W = \frac{4bd^2}{L} R$.

So it is evident that failure of flexure would theoretically take place at the points of support and not at the center of the span.

In applying the formula $W = \frac{2bd^2}{L} R$ to the case of a slab spanning a breast or other mine opening, the weight of the overlying material will be taken at 108 lb. per cubic foot, and the depth of the opening below the surface will be designated by d' in feet.

We will then have $W = \frac{108 Ld'}{12}$, which would be the loading of the slab with a breadth of 1 ft. Substituting this value of W in the equation $W = \frac{2bd^2}{L} R$ and simplifying, we obtain:

$$d^2 R = \frac{3}{8} L^2 d'.$$

By taking the width of the opening, L , as 24 ft. and 100 lb. per square inch as the value of R , the formula becomes $d = 17.63\sqrt{d'}$,

which becomes $d = 1.47 \sqrt{d'}$ for values of d in feet. For 120 lb. per cubic foot this becomes $d = 1.55 \sqrt{d'}$, so, for convenience in use and without appreciable error, this is taken as $d = 1.50 \sqrt{d'}$.

From this formula Table IV has been prepared, which gives the thicknesses of rock for depths ranging from 10 to 300 ft. and 24 ft. width of chambers.

In the use of the formula derived for determining the minimum safe thickness of rock over mine openings for various depths below the surface, consideration must be given to a number of conditions, the more important of which are:

1. Nature of the top immediately above the coal seam, its comparative strength and liability to disintegration upon exposure to the atmosphere.

To compensate for this condition, the allowance to be made can be determined by observing the nature of the top in places that have been mined nearest to the location under consideration. Every vein and section may present different conditions, so that allowances varying from 1 to 10 ft. may be found necessary.

2. Nature and thickness of the vein, the ability of the pillars to resist squeezing, and the liability of disturbance to the overlying strata, due to caving or squeezing in underlying veins.

To compensate for these conditions is more difficult, for the reason that a squeeze will crack the overlying strata and admit water in varying quantities, although the thickness of rock cover may be far in excess of any requirement against caving. Consequently, there would be no necessity for making any allowances in the rock cover thickness for these conditions. It would, however, be advisable to compensate against squeezing in the vein, over which the rock cover is being established, by mining with a larger factor of safety.

3. Probable errors in relative vertical location of top of rock and mine workings.

To compensate for possible errors in vertical locations an allowance of 5 ft. in the thickness of the rock should be made. The compensation for this condition will depend upon the verification of surveys on the surface with those in the mine, and unless the elevations are absolutely verified an allowance of at least 5 ft. in the rock cover should be made.

4. Possibility of the existence of deep gorges and pot holes.

To compensate for possible gorges and pot holes an allowance of 35 ft. should be made.

The thickness of rock cover required for 24 ft. width of chambers to support safely the overlying strata at various depths is graphically illustrated in Fig. 10.

The curve *A* is derived from the equation $d = 1.5 \sqrt{d'}$ and repre-

sents the minimum thicknesses of rock where all conditions are absolutely known. This curve could rarely, if ever, be considered in practice, for the reasons before given.

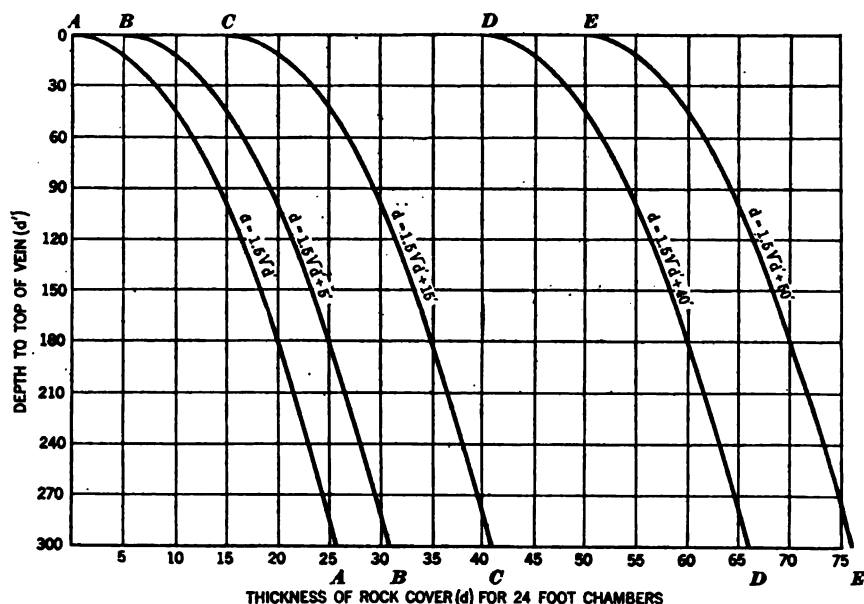


FIG. 10.

TABLE IV.—*Minimum Thickness of Rock for Various Depths below the Surface, over Chambers 24 Ft. Wide*

Depth d', Feet	Thickness of Rock d, Feet	Depth d', Feet	Thickness of Rock d, Feet
10	4.74	160	18.97
20	6.71	170	19.55
30	8.21	180	20.13
40	9.48	190	20.67
50	10.61	200	21.21
60	11.62	210	21.74
70	12.55	220	22.25
80	13.41	230	22.75
90	14.23	240	23.23
100	15.00	250	23.72
110	15.73	260	24.18
120	16.43	270	24.65
130	17.10	280	25.10
140	17.75	290	25.54
150	18.37	300	25.98

Curve *B* represents the minimum values for the thickness of rock cover *d* after making an allowance of 5 ft. for probable errors in relative elevations, and would only be used when the top rock of the vein is hard, strong sandstone or other rock of equivalent strength and durability. This curve could only be considered when the contour of the top of rock is absolutely known and there is no possibility of the existence of pots or deep eroded gorges.

Curve *C* represents the minimum values for the thickness of rock cover after making an allowance of 10 ft. for poor top in addition to the 5 ft. for probable errors. This curve could only be considered when other conditions as required for curve *B* are known.

Curves *D* and *E* represent the values for the thickness of rock cover after making an allowance in each case of 35 ft. for pots and gorges in addition to the allowances made for curves *B* and *C* respectively. Curve *D* would be used when the top is hard sandstone, and curve *E* when the top is soft and disintegrating.

Conditions may warrant the reduction of the allowances as used in determining curves *D* and *E*, but unless these conditions are positively proved, it would not seem advisable to mine with less rock cover than is represented by curve *D*.

The practices in mining under wash of various depths have not been very consistent, and it is with the idea of standardizing future practice and at the same time securing more safety and efficiency in mining under these conditions that this paper is presented.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Depreciation as Applied to Oil Properties

BY PHILIP W. HENRY, NEW YORK, N. Y.

(New York Meeting, February, 1915)

THERE is a difference of opinion among engineers on the subject of depreciation in general, and still more on its application to any given case. The committee which was appointed by the American Society of Civil Engineers to make a report on Valuation of Public Utilities, states "There is no subject connected with valuation about which there are more diverse views than those relating to depreciation;" and in the discussion which took place upon the presentation of the preliminary report of this committee in January, 1914, that part of the report dealing with depreciation called forth more discussion than any other. From this discussion it was evident that the word "depreciation" was not always used in the same sense, so it is quite important that some definition be adopted. As this paper relates to the proper method of treating depreciation by corporations or individuals operating oil properties, both of which must make a return of annual net income to the government in connection with the income tax, it will be in order to adopt the definition of the Internal Revenue Bureau. Under date of Mar. 29, 1910, the Commissioner of Internal Revenue of the Treasury Department stated "Deduction on account of depreciation of property must be based on lifetime of property, its cost, value and use." Again, on Form 1035, "Return of Annual Net Income" for miscellaneous corporations (Section 2, Act of Congress approved Oct. 3, 1913), a deduction from gross income is allowed for "Total amount of depreciation for the year." This item is defined more particularly as follows:

"The amount claimed under Item No. 5 (b) for depreciation should be such an amount as measures the loss which the corporation actually sustains during the year in the value of buildings, machinery, and such other property as is subject to depreciation on account of wear and tear, exhaustion, or obsolescence. The amount taken credit for on this account in order to be allowable should be so entered on the books as to constitute a liability against the assets of the corporation. The amount claimed under this item should not cover losses in the value of stocks and bonds. Decrease in the book value of securities owned, so far as such decrease represents a decline in the actual value of such securities, should be deducted under item 5 (a) of the return.

"Where depreciation of physical property is made good by renewals, replacements,

repairs, etc., and the expense of such renewals, replacements, repairs, etc., is charged to the general expense account, no deduction for depreciation can be made in the return of annual net income. Where a depreciation reserve is set up, all renewals and replacements must be charged to such reserve and the addition to this reserve each year must be a fair measure of the loss which the corporation sustains by reason of the depreciation of its property."

Here the Internal Revenue Bureau recognizes that depreciation includes wear and tear, exhaustion, and obsolescence, and that in arriving at the amount to be charged off each year there must be taken into account the lifetime of the property, its cost, value, and use. The Bureau also indicates the difference between depreciation and ordinary wear and tear—more or less uniform in amount each year—which is generally carried as a regular operating account under the designation of "repairs and renewals" or some similar title. The Bureau also refers to the setting up of a "depreciation reserve," which is undoubtedly the best way to treat depreciation on the books, where it is carried as a credit account, although on the balance sheet it may be shown either on the credit side as a liability or on the debit side as a deduction from one or more of the property accounts. This account should be credited annually (or preferably monthly) with an amount sufficient to provide a fund to take care of the cost of deferred or final renewals, and in the case of an oil property, to provide a fund to balance the depletion of oil lands. As depreciation reserve is both a debit and a credit account, against it should be charged the cost of deferred and final renewals when made. By so doing the annual net income is not affected by large expenditures made in any one year for new plant to replace that which has been discarded on account of deterioration, inadequacy, or obsolescence. Depreciation reserve should also be charged with the cost of such new oil lands as are necessary to maintain the original value of oil lands. Theoretically, the amount standing in depreciation reserve at any given time plus the value at that time should equal the original value; but seldom does this condition obtain, as it is exceedingly difficult to determine the proper amount to set aside each year in depreciation reserve. Strictly speaking, the only way to arrive at this amount is to make a careful valuation each year, and debit or credit depreciation reserve accordingly. Manifestly such yearly valuation is impracticable, and it is more convenient to assume a lifetime for each item making up the property, and it is this method which is recommended by the Internal Revenue Bureau. It is therefore evident that the annual allowance for depreciation cannot be determined with mathematical accuracy, as both the lifetime and the condition of any property or of its component parts are only estimates, on which no two engineers will agree in detail. It would therefore seem that, particularly as applied to oil properties, any discussion as to the relative merits of the various methods of setting aside an annual amount for depreciation would introduce a

refinement not warranted by conditions. Mention, however, will be made of the three principal methods. By the sinking-fund method a certain equal percentage is set aside each year, which, being compounded annually at a given rate of interest, will at the end of the assumed lifetime amount to the original value. By the equal-annual-payment method certain increasing percentages are set aside each year, which, without interest, will at the end of the assumed lifetime amount to the original value. By the straight-line method a certain equal percentage is set aside each year, which, without interest, will at the end of the assumed lifetime amount to the original value. For oil properties, with an inevitably decreasing production (after maximum is reached), it would seem that some method should be adopted by which decreasing percentages should be set aside each year for depreciation. Such a method would more nearly equalize the depreciation charge per barrel of production. On the other hand, it is a question whether, on account of the higher selling price which generally obtains as an oil field is depleted, the later years cannot stand a higher depreciation charge per barrel. In view of these uncertainties it would seem that the straight-line method is simplest and best, for, owing to the extreme difficulty in arriving at the lifetime of the various items making up the property, the greater exactness called for by the other methods is unnecessary. No matter what method is adopted and no matter how conscientiously and intelligently applied, in all probability the experience of a few years will show that the amount in depreciation reserve is too large or too small, and that the annual rate of depreciation should be changed accordingly. For this reason it is advisable to adopt at proper intervals what has been termed the actual-inspection method, by making a physical valuation of those parts of the property susceptible to depreciation, in order to determine whether or not the amount in depreciation reserve is equal to the original value less present value, as it should be—leaving out of consideration the effect which appreciation and other elements connected with valuation may have on the subject, and assuming that original cost and original value are equivalent terms.

While the proper treatment of depreciation is required by the government for purposes of taxation, it is of still greater importance to the stockholder, for only then does he know that his investment is being kept up to its original value, and that the dividends paid, whether 10, 50, or 100 per cent., are really earned. Whether depreciation reserve is invested in outside securities, as may be advisable in the declining days of an oil property, or whether invested in the property itself for betterments and additions, is immaterial. If properly set aside, the stockholder knows that out of earnings each year has been provided a fund, not applicable to dividends, which maintains his investment at a constant value, and which, when the oil is exhausted, will be sufficient, together with the value of the remaining property, to pay him back his principal. This of course repre-

sents an ideal condition, seldom realized in practice; for while it is very easy to enunciate principles, it is most difficult to apply them to the particular case in hand. Such application, however, will now be attempted.

First of all, the lifetime of every item entering into the property should be established, and a depreciation reserve set aside annually for each item based upon such lifetime, or an average lifetime established for the entire property. An oil property, the lifetime of which as a whole, or in part, is so uncertain, may be considered as being made up of three classes, viz.: oil lands, field equipment, and wells. In field equipment should be included tanks, reservoirs, pipe lines, loading racks, water system, power plants, structures of various kinds, and miscellaneous items belonging to the field as a whole rather than to the individual well. In wells should be included the well itself and those appurtenances, the lifetime of which depends more upon the individual well than upon the field. This simplifies the matter in that only the average lifetimes of the oil lands, the field equipment, and the individual well need be established, thus reducing book entries to a minimum.

It is manifestly beyond the province of this paper to select any particular oil property for the purpose of illustrating the general principles of depreciation, and even were such property thus selected, the application of those principles to that particular case would have no more value when applied to another property, than if the field as a whole or even a State as a whole, such as California, were selected for purposes of illustration. It is obvious that a rate of depreciation established for the property containing the Lakeview gusher, for example, would be of little value when applied to another property in the same field, or even to an adjoining property. For this reason the State of California as a whole has been selected for purposes of illustration, and it is believed that the end in view will be better served than if some specific property were taken. In establishing the lifetime of the oil fields of California as a whole or in part, the best recourse is to the United States Geological Survey, which for years has made a careful study of the various oil fields of the United States and publishes a report every year upon the subject. For California the Survey has estimated that the original contents of the probable oil lands were somewhere between 5,000,000,000 and 8,500,000,000 barrels, or from one-third to one-half of the entire probable production of the United States. There have been produced in California as a total to date some 750,000,000 barrels, leaving still in the ground from 4,250,000,000 to 7,750,000,000 barrels. At the present rate of production, 100,000,000 barrels yearly, it will take from $42\frac{1}{2}$ to $77\frac{1}{2}$ years to exhaust the oil, with an average of 60 years. It must be remembered, however, that within the past 10 years the annual production of the State has trebled and within the last five has doubled, the production in 1904 having been 33,427,273 barrels and in 1909, 54,433,010 barrels.

On account of this constant annual increase in production, as well as the danger from water intrusion, which seriously threatens some fields of the State, it would seem prudent to adopt a lifetime of not more than 25 years for the field; and assuming that the oil lands owned by the hypothetical company under consideration are up to the average, a depreciation rate of 4 per cent. per annum is indicated upon their cost, considering in this case cost and value as equivalent terms. This annual rate, being predicated upon total extinction of value in 25 years, will be modified in accordance with the value of the land after the oil is extracted. For example, should the land after the oil is extracted be worth 25 per cent. of its original value, the yearly depreciation rate will be 3, instead of 4 per cent. For purposes of illustration, however, the land, upon exhaustion of the oil, will be considered as of no value, which is not far from actual conditions in some parts of California. In order to apply the principle, some cost (value) of oil lands must be assumed, and for purposes of illustration a cost of \$500 and \$1,000 respectively per barrel of daily production will be used. On a production of 1,000 barrels per day, the cost will be \$500,000 and \$1,000,000 respectively, with yearly depreciation charges of \$20,000 and \$40,000 respectively, which on a yearly production of 365,000 barrels amounts to \$0.055 and \$0.110 respectively per barrel. As depreciation charge on oil lands takes the place of royalty paid by companies operating under lease, the results obtained are not unreasonable.

In order to establish the lifetime of field equipment, that of each component part must be estimated and a general average established. Some of the equipment may last the lifetime of the field. Other parts may have to be renewed every five years, so that here again each property must be studied by itself. Assuming an average lifetime of about 15 years, which seems not unreasonable, an annual depreciation rate of 7 per cent. is indicated. In applying this rate to a company of which I have knowledge, with a field equipment costing \$150,000 per 1,000 barrels of production daily, the annual depreciation charge will be \$10,500, or \$0.029 per barrel. In applying to another company of which I have knowledge, with a field equipment costing \$270,000 per 1,000 barrels of production daily, the annual depreciation charge will be \$18,900, or \$0.052 per barrel.

In regard to the lifetime of an individual well, that again is a local question and must be determined in accordance with facts to be ascertained in each particular field. There are wells in the Kern River district which have been in operation for 15 years and are still producing a fair percentage of their original production, but this is an exceptional field for persistence. In this determination may also be considered the cost of drilling additional wells in order to maintain production at a constant figure. This is frequently carried as a regular operating expense, similar to repairs and renewals, but as depreciation, as herein defined, is a function of capital account rather than of annual production, and as such expense may vary greatly from year to year, it is a question whether drilling

to maintain production should not be charged into capital account, and depreciated along with the other wells. Such a treatment naturally increases the annual percentage of depreciation, but as this rate is, after all, a matter of estimate and subject to change as new conditions develop, it seems preferable to treat drilling to maintain production as a charge against capital rather than as an ordinary operating expense. Giving due consideration to this element in the problem, 10 years will be assumed as the average lifetime of a well and its appurtenances, taking also into account dry holes, having no lifetime, on the one hand, and those wells which may produce for 20 years or more on the other. Applying this annual depreciation charge of 10 per cent. to one company of which I have knowledge, with an investment of \$190,000 in cost of drilling wells per 1,000 barrels of production daily, and to another with \$260,000 so invested, there will be a yearly depreciation charge of \$19,000 and \$26,000 respectively, amounting to \$0.052 and \$0.071 respectively per barrel.

Summing up these depreciation charges we arrive at the following results per barrel of production:

	Per Barrel
Depreciation on oil lands (royalty).....	\$0.055 to \$0.110
Depreciation on field equipment.....	0.029 to 0.052
Depreciation on wells and appurtenances.....	0.052 to 0.071
Total depreciation.....	\$0.136 to \$0.233

These figures must not be regarded either as the minimum or the maximum which may obtain in any given operation, and particularly is this true of oil fields outside of California, and where the persistence of the wells may be greatly different. These figures, however, serve to call attention to the important bearing which a proper charge for depreciation has upon the cost of producing oil in a State where, during the past few years, prices at the well have ranged from 30c. to 85c. per barrel, depending upon the gravity of the oil and the location of the field. It must also be remembered that, in addition to this charge for depreciation, there will be an item for renewals and repairs, charged directly into the cost of operating, which may range from 3c. to 5c. per barrel. It is therefore evident that the actual cost of producing oil in California, no allowance being made for interest on the investment, must in many cases approximate the selling price, as shown in the following statement, based upon experience:

	Per Barrel
Pumping.....	\$0.04 to \$0.05
Miscellaneous field expense.....	0.04 to 0.06
Repairs and renewals.....	0.04 to 0.05
General expense.....	0.02 to 0.04
Total direct cost.....	0.14 to 0.20
Depreciation as above.....	0.14 to 0.23
Total cost.....	\$0.28 to \$0.43

That a proper depreciation charge may be equal to, or greater than, the direct cost per barrel will doubtless surprise many investors in oil properties who consider only immediate expense and immediate profit without regard to the safety of the principal invested.

To engineers, however, who have had the responsibility of operating oil properties, these figures will not be surprising; for in his interesting and valuable paper on Present Conditions in the California Oil Fields, presented to the Institute at its San Francisco meeting in October, 1911, M. L. Requa states:¹

"It is exceedingly to be regretted that the oil-producers of California, as a whole, do not apparently realize the real cost of production. . . . From territory of, say, 2,500 ft. depth, total costs will approximate from 30 to 35 cents per barrel. For direct production—i.e., pumping, cleaning, and pulling—10 cents per barrel may be safely assumed. For maintenance of surface-equipment and rigs, 4 cents is a conservative estimate. For exhaustion of oil-land, and redemption of capital, from 6 to 10 cents must be reckoned; and for drilling to maintain production, 12 cents is not excessive. These figures make a minimum of 32 cents and a maximum of 36 cents."

While Mr. Requa has arrived at his estimated cost by a different method of accounting, the average is practically the same, and it would be most instructive to have the opinion of other engineers on this subject. To draw attention to the proper application of depreciation to oil properties, and to its importance as an element in the cost of producing oil, is the purpose of this paper, and it is hoped that a full discussion will be made by those members of the Institute whose experience in the various oil fields of the world will be of great value, not only to engineers, but also to all those interested in the development of oil properties.

To sum up, my conclusions are:

That there should be set aside each year, out of earnings, a depreciation reserve, sufficient in amount to keep the property at a constant value, by providing a fund to take care of extraordinary wear and tear, inadequacy, obsolescence, and exhaustion.

That ordinary wear and tear should be charged directly into operating.

That depreciation reserve, being both a debit and a credit account, should be charged with extraordinary repairs and with replacements as made.

That depreciation reserve should not be applicable to the payment of dividends, but only for the repayment of principal.

That depreciation reserve may be invested in the property itself or in outside securities.

That the principles used in setting aside depreciation reserve should be those outlined by the Internal Revenue Bureau, quoted in the beginning of this paper.

¹ *Trans.*, xlii, 841 (1911).

That in the application of these principles it is necessary to determine the lifetime of each item making up the property, but that, in view of many uncertain factors pertaining to oil properties, it is sufficient accurate to divide depreciable property into three classes—oil lands, equipment, and wells—and determine the average lifetime of each class.

That these average lifetimes, with due consideration of remaining value upon exhaustion of oil, will determine the annual percentage charge on these three items standing in capital account, the procedure which should be credited to depreciation reserve.

That these percentages, being based upon estimates, are subject to change, and should be verified at proper intervals by making actual calculations in order to determine whether the amount in depreciation reserve plus present value equals original value, as it should—neglecting elements of valuation, and using cost and value as equivalent terms.

That in establishing lifetimes each property must be studied by itself, and different annual percentages must necessarily be adopted for different properties on account of the great variation in conditions.

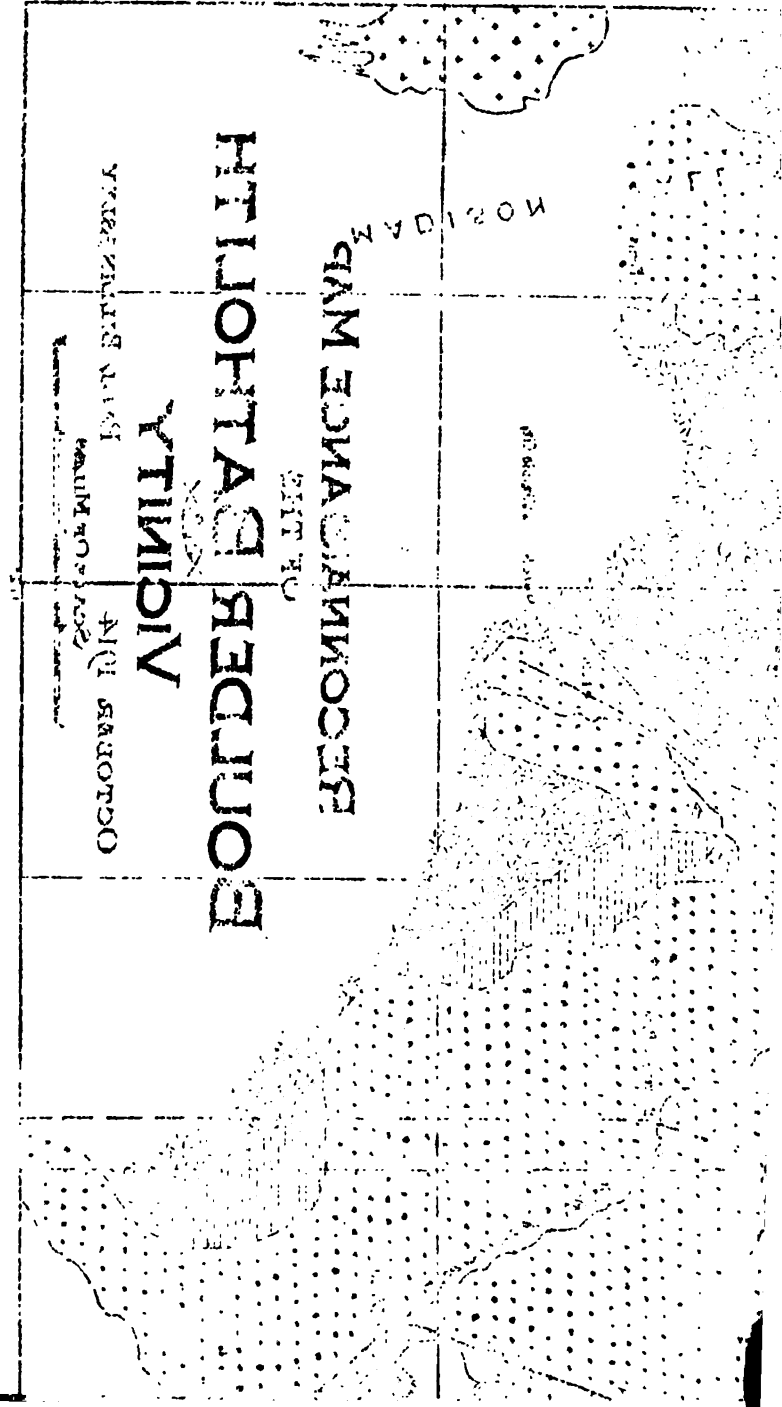
That it would, however, be advantageous to establish for each field average percentages which could be used until such time as the characteristics of the different properties in the field are known.

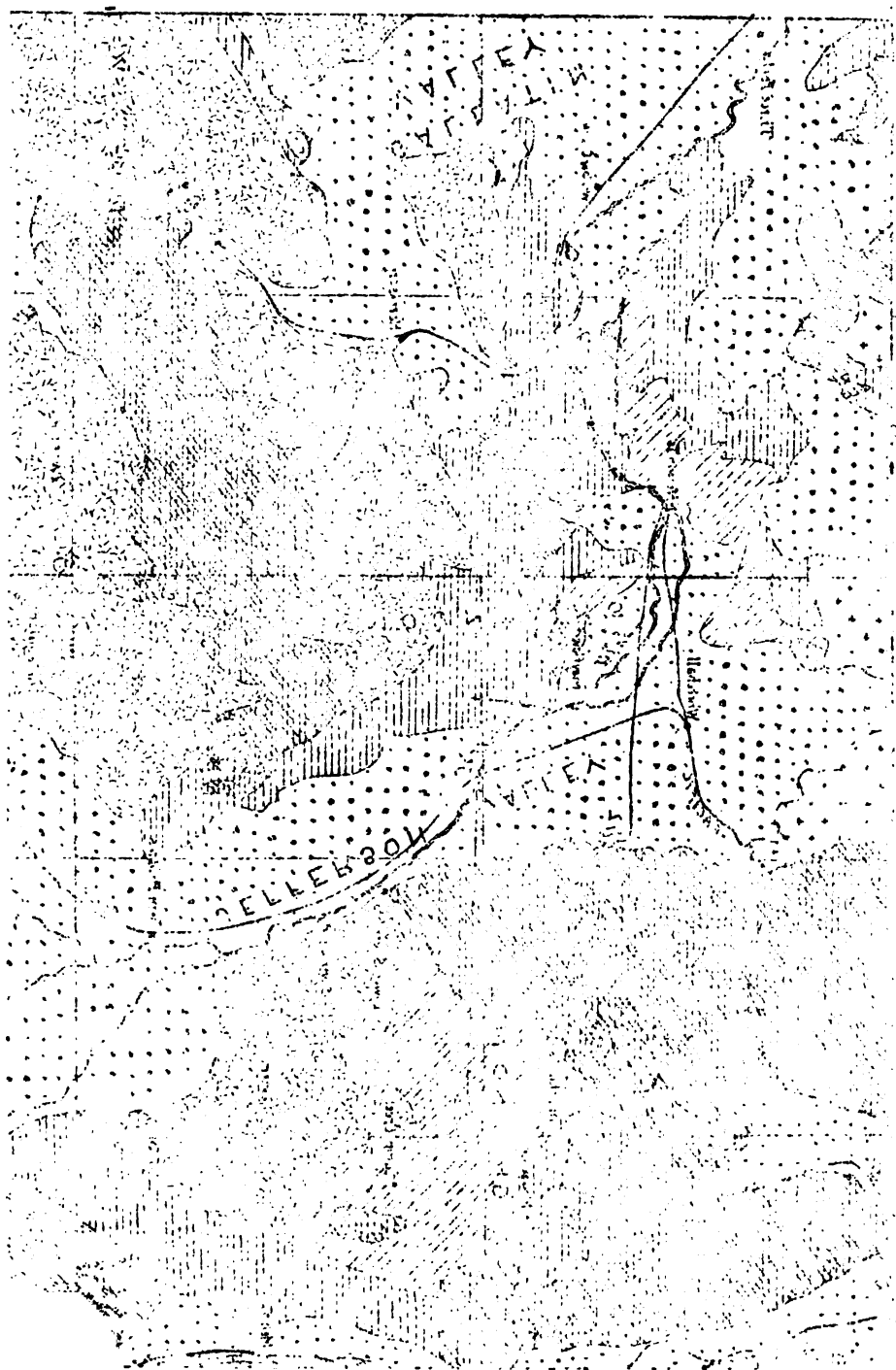
That for California such average annual percentages should be 10 per cent. on cost of oil lands, 7 per cent. on cost of field equipment and 5 per cent. on cost of individual wells and appurtenances, but that these percentages should be changed in accordance with the development and characteristics of the property under consideration.

That in the oil fields of California depreciation cost may equal or exceed direct cost of producing oil per barrel.

These conclusions are not intended to be dogmatic, but rather for the purpose of stimulating discussion, for only by combining the experience and judgment of many engineers can depreciation be correctly applied to oil properties.

ПЯТАЯ ГЛАВА ОДНОГО ИЗ ПОВЕЩАНИЙ Л. П. ПУШКИНА





DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Boulder Batholith of Montana¹

BY PAUL BILLINGSLEY, BUTTE, MONT.

(New York Meeting, February, 1915)

THE term Boulder batholith was first applied in 1897 by W. H. Weed² to the extensive mass of granite in western Montana within whose borders occur the ore deposits of Butte. In a general way this was known to extend from Helena on the north to the Highland Mountains near Butte on the south, and to be relatively narrow in its east-west diameter. Its intrusive relations to the surrounding sedimentary rocks, and its probable Tertiary age, were described by Weed, but in many ways his statements, based upon incomplete data, have required revision. Most notable of the errors of this description of the batholith were the failure,³ since corrected, to recognize the intrusive contact of the granite with the capping of andesite, and the complete absence of any reference to the many extensions of the granite beyond the somewhat narrow limits of its largest exposure.

¹ The field observations upon which this paper is based have been made during the past five years by J. A. Grimes, of Butte, and the author. Over 300 thin sections, prepared and examined by A. H. Smith, of Gateway, B. C., have assisted in determining doubtful points. Reports on mines by Reno H. Sales, F. A. Linforth, and Murl Gidel, of Butte, have been freely drawn upon.

The accompanying geologic map is based primarily upon original field work. A considerable area thus covered has since been mapped in publications of the U. S. Geological Survey, and these maps, where of greater accuracy, have been incorporated. A map of the Helena region accompanying *Professional Paper No. 74* (Ore Deposits of the Butte District, by W. H. Weed) has required revision, but the more recent map of Knopf covering the same area has been drawn upon freely. Stone's map of the Elkhorn Mountains has likewise helped to fill a gap, while the wonderfully accurate map accompanying *Professional Paper No. 78* (Philipsburg Quadrangle, by W. H. Emmons and F. C. Calkins) has been used without modification. The only criticism possible on this map is the employment of divers and contrasting colors in representing granitic rocks of common age and origin and with but slight textural and mineralogical differences.

² *Butte Special Folio*, U. S. Geological Survey (1897). Also, Granite Rocks of Butte, Mont., and Vicinity, *Journal of Geology*, vol. vii, No. 8, p. 737 (Nov.-Dec., 1899).

³ *Butte Special Folio*, p. 1 (1897).

These outlying areas of the granite, however, did not escape the attention of other and earlier observers, and in the geological reports of the Hayden expeditions,⁴ as well as the descriptions of the early quartz mines of the State compiled by R. W. Raymond,⁵ references to the granite abound. A common error among these early writers was the general grouping of the granite with the gneiss and schist frequently found, as a complex basal system of Archean age, and for this reason their observations, while remarkably accurate in detail, have failed to bring into prominence the widespread extent of the intrusive granite. Absence of detailed study of the gaps which separate these areas from the Boulder batholith of Weed has helped to perpetuate the impression of the isolation and well-defined limits of that mass.⁶

More recent field work by the Geological Survey⁷ has, however, demonstrated the intrusive character and probable Tertiary age of large granite areas both west and east of the original batholith, and while many of the largest outlying masses have not been described, their general relations and character are sufficiently well known to include

⁴ F. V. Hayden: *U. S. Geological Survey of the Territories* (Washington, 1867, 1868, 1869, 1872, 1873).

⁵ J. Ross Browne: *Mineral Resources of the United States* (Washington, 1866, 1867, 1868).

Also, R. W. Raymond: *Mineral Resources West of the Rocky Mountains* (Washington, 1870 to 1875, inclusive).

⁶ In the past year Prof. A. C. Lawson, of Berkeley, has advanced the hypothesis that the Boulder batholith is in fact a laccolith, occupying a great synclinal basin, and upon this hypothesis has based theories of meteoric origin for its ore deposits. Professor Lawson's paper is based largely upon a review of the literature of the region, and many of his essential premises are not confirmed by field evidence. It can, for example, be definitely shown that the "synclinal basin" is a portion of Rocky Mountain structure several geologic ages earlier than the granite, and is in no sense due to the intrusion of the latter; that the granite has in no place penetrated between the andesite capping and the underlying sediments, but has truncated the sediments below, and only at high points has reached the andesite; and that magmatic chalcoppyrite, rather than being indicative of the basic border of the intrusive mass, is closely associated with the siliceous injections of the later segregations. It is difficult to accept Professor Lawson's paper as a candid analysis of the problem.

⁷ R. W. Stone: *Geologic Relation of Ore Deposits in the Elkhorn Mountains, Montana, Bulletin No. 470, U. S. Geological Survey*, pp. 75 to 98 (1911).

W. H. Weed: *Geology and Ore Deposits of the Butte District, Montana, Professional Paper No. 74, U. S. Geological Survey* (1912).

A. Knopf: *Ore Deposits of the Helena Mining Region, Montana, Bulletin No. 527, U. S. Geological Survey* (1913).

J. B. Umpleby: *Geology and Ore Deposits of Lemhi County, Idaho, Bulletin No. 528, U. S. Geological Survey* (1913).

W. H. Emmons and F. C. Calkins: *Geology and Ore Deposits of the Philipsburg Quadrangle, Montana, Professional Paper No. 78, U. S. Geological Survey* (1913).

A. N. Winchell: *Mining Districts of the Dillon Quadrangle, Montana, and Adjacent Areas, Bulletin No. 574, U. S. Geological Survey* (1914).

them also in the group. Growing knowledge of the structure and nature of the rocks filling the gaps which appear on the map between the several granite areas, leads also to the belief that these tracts, isolated on the surface, are in reality but the many exposures of a single mass.

The original Boulder batholith remains the largest connected outcrop of this mass, which may be tentatively called the Montana batholith; but large exposures are now recognized to the south and west, while numerous smaller areas indicate its presence at no great depth for a considerable distance to the north and east of the old limits. Gauged by these standards, the granite batholith of Montana has a width of 75 miles, from Philipsburg to Elkhorn, and a length, from Marysville to Dillon, of well over 100 miles.

Batholith in Mountainous Section of Montana

This area is entirely within the mountainous portion of the State, falling between the Rocky Mountain main range on the east and the parallel Bitterroot system on the west. The granite is thus intimately associated in occurrence with the great mountain folds and faults of the Montana cordilleras. These may be structurally divided into several belts, paralleling the northwest course of the ranges.

Most easterly is an outlying zone of gentle folds, which find topographic expression in the Little Belt and Big Snowy mountains. Next on the west is the Rocky Mountain belt proper, carried from the Canadian line to the Yellowstone by the Livingstone, Main, and Big Belt ranges—a belt of closely folded and faulted rocks overriding the eastern zone along great thrust faults. The western border of this zone is indefinite; the folds decrease in intensity, and the mountains drop off into the low levels of the great valleys. These valleys, the Missouri-Jefferson on the east, and Blackfoot-Deer Lodge on the west, beginning on the flanks of the Rocky Mountain belt, and converging southward, surround a rugged plateau, whose area coincides in general with the extent of the main mass of the Montana batholith. To the west, another zone of thrust faulting inaugurates a third mountain belt that includes the complex ranges beyond the Deer Lodge and Melrose valleys (Fig. 1).

The several belts into which the Montana cordilleras may thus be topographically separated coincide with zones of geologic structure. Each belt is in a large way a complete earth fold, anticlinal to the east, synclinal to the west, with Algonkian rocks exposed by the erosion of the former division, and Cretaceous remaining in the basins of the latter. Thrust faulting has caused the anticlines to override the adjoining eastern synclines, and these intensely folded and faulted zones serve to separate the mountain belts.

The time relation of the igneous rocks of the Montana batholith to



this general mountain structure can be fixed with considerable certainty, and is summarized in the following table:

1. Middle Cretaceous.—Main Rocky Mountain folding, and formation⁸ of the large earth folds in northwesterly direction.
2. Middle-Upper Cretaceous.—Extensive erosion and beveling of folds. Deposition of terrestrial and shore deposits to east.
3. Upper Cretaceous.—Andesite eruption. Deposition of extrusive lavas and breccias upon eroded surface to west, and formation of tuffs and andesitic sediments to east.
4. Upper Cretaceous(?).—Local intense erosion and formation of coarse andesite conglomerates.⁹
5. Upper Cretaceous(?).—Thrust faulting along northwest lines, and local intensification of folding. Andesitic sediments are included in this movement.¹⁰
6. Eocene(?).—Intrusion of Montana granite.
7. Eocene.—Extensive erosion. Development of peneplain and well-adjusted river system. Early rhyolite.
8. Oligocene.—Normal north-south faulting, and local accumulation of gravels. Erosion of peneplain.
9. Miocene.—Further accumulation of river deposits. Later rhyolite and dacite. Same conditions probably extended through the Pliocene period.

The geological expression of this series of events can be seen in a few critical sections. The first (Fig. 2), along the general line of the Northern Pacific railroad between Helena and Garrison, shows the unconformity between the early folding and the andesite, and the antecedence of the structure to the granite intrusion. The second (Fig. 3), about 30 miles to the south and east, in addition places the thrust faulting after the andesites and before the Tertiary gravels. The coarse andesite conglomerate at Maudlow, near the eastern end of the section, could have been derived from no other source than the lavas of the Elkhorn Mountains. It is, therefore, of later age than the pre-andesite folding, but with the conforming Cretaceous sandstones it has been folded and overturned by the action of the thrust zone to the west. The Lost Creek sections (Figs. 4 and 5), a few miles north of Anaconda, prove the granite to be subsequent to the thrust faulting, while the two remaining sections (Figs. 6 and 7) show the relation of the two rhyolite series to the granite, the gravels, and the erosion surfaces. These are but typical examples. Instances might be multiplied, but the concordance of all evidence found and the absence of conflicting facts leave but little doubt of the essential accuracy of the above tabulation.

The intrusion of the granite was thus a relatively late event in the development of the Montana cordilleras, and the main features of structure had already been developed. The present topography, on the

⁸ *Bulletin No. 470, U. S. Geological Survey*, p. 78 (1911).

⁹ Type locality, hill top southeast of Garrison, where coarse cemented andesite conglomerate lies unconformably upon Kootenai Cretaceous sandstones.

¹⁰ Type locality, Maudlow, Broadwater County.



SECTION A

FIG. 2.—GEOLOGICAL SECTION ALONG THE LINE OF THE NORTHERN PACIFIC RAILROAD BETWEEN HELENA AND GARRISON.



SECTION B

FIG. 3.—GEOLOGICAL SECTION OF COUNTRY 30 MILES SOUTH AND EAST OF FIG. 2.

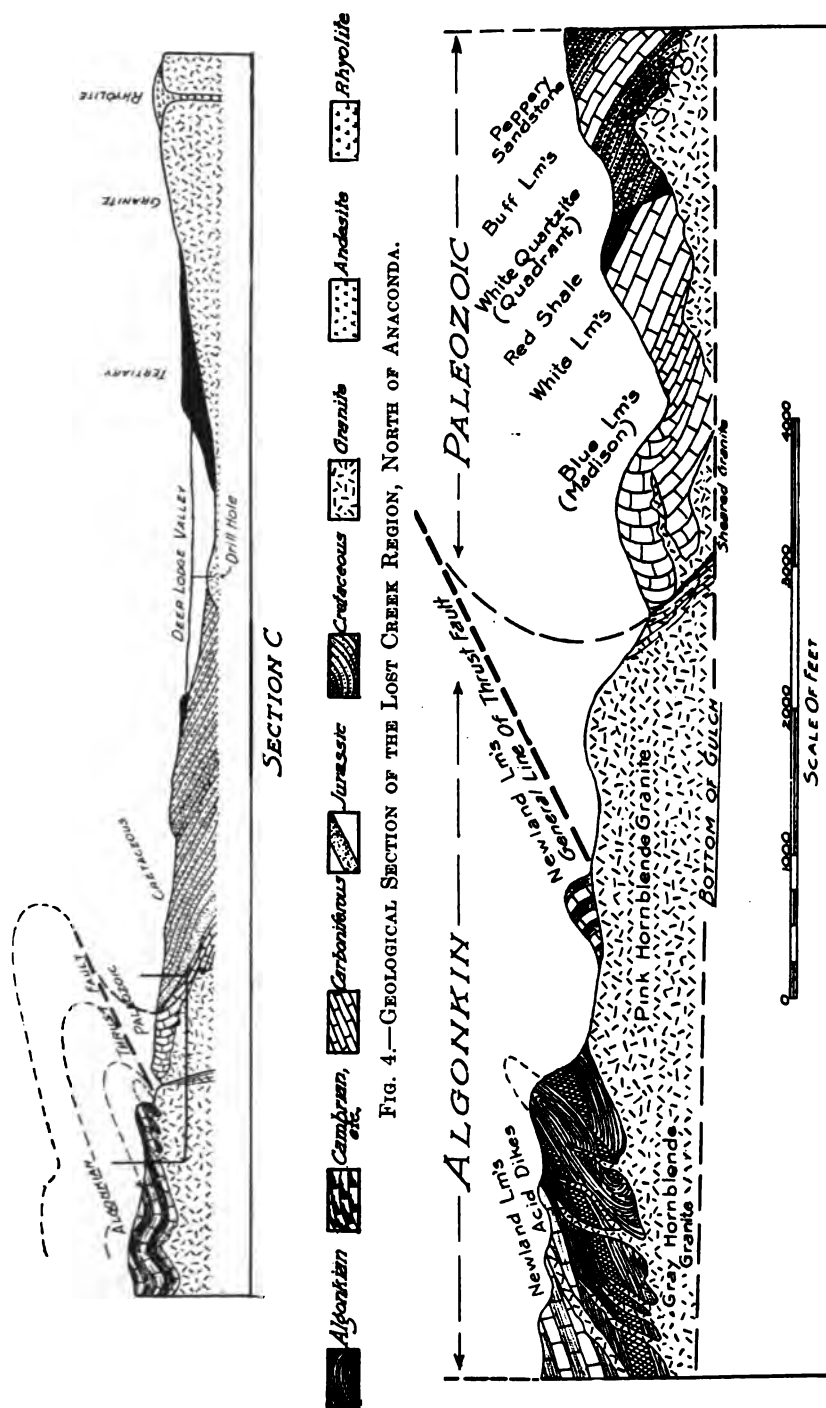


FIG. 4.—GEOLOGICAL SECTION OF THE LOST CREEK REGION, NORTH OF ANACONDA.

FIG. 5.—DETAIL OF SECTION SHOWN IN FIG. 4.

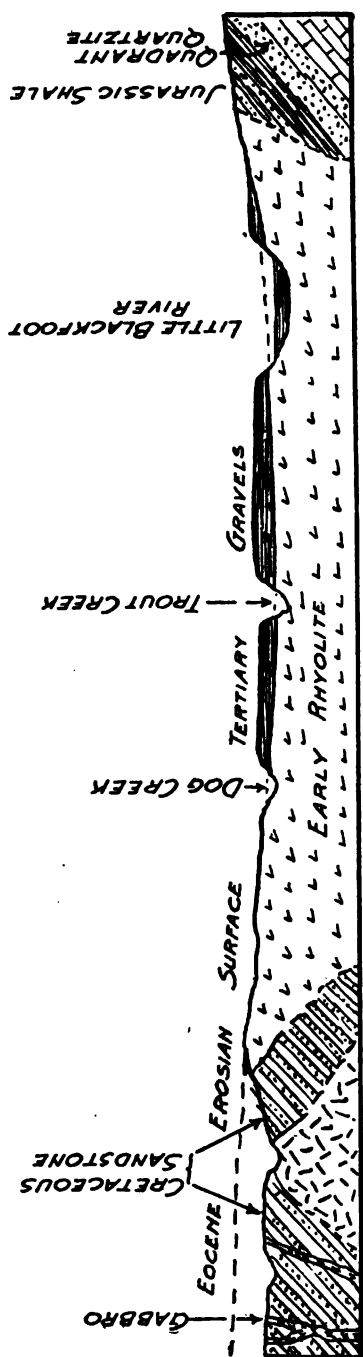


FIG. 6.—SECTION SHOWING OCCURRENCE OF EARLY RHYOLITE, NEAR AVON.

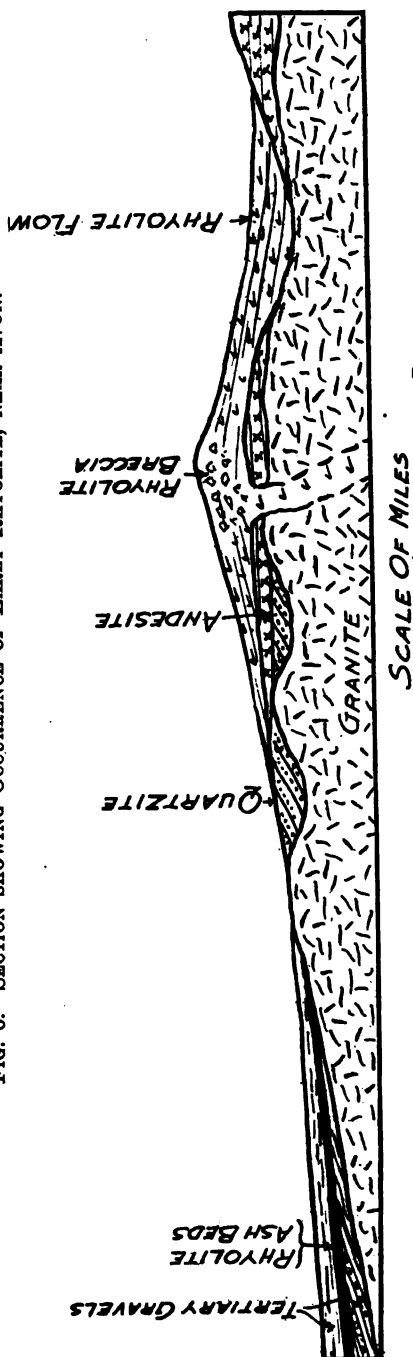


FIG. 7.—TYPICAL SECTION, EAST SIDE OF DEER LODGE VALLEY.

other hand, is of post-granite origin, and the presence of the batholith has determined in no small degree the great valley lines of the mountain region.

Granite Intruded as Dome-Shaped Mass

From fragments remaining of the original cover, and from the position of contacts on the edges of the several granite areas, it is possible to fix within general limits the form of the original top of the intrusive mass. This may be roughly described as a double dome, with high points east of Butte and north of Basin separated by a deep trough. From these crests the granite surface, irregular in detail, slopes gently to the east, north, and west, and is exposed at remote points wherever erosion has

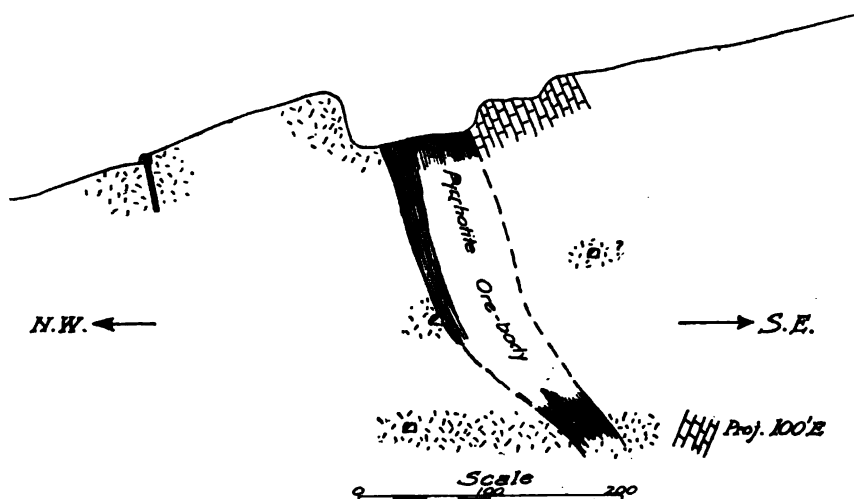


FIG. 8.—GOLDEN CURRY, JEFFERSON COUNTY. AFTER RENO H. SALES.

reached low elevations. It is evident that with increased denudation more and more of the granite is being exposed, and its frequent appearance, with normal texture and composition, at low points in the peripheral gulches points unmistakably to its general presence at no great depth. Granite cappings on outlying hills are entirely absent, and there is no evidence of granite overlying the earlier rocks except in the form of local dikes or sills. At many points on the periphery of the main granite area the contact is exposed in deep gulches or in mine workings. Later faulting at many points confuses the contact, but the unbroken downward extent of the granite and the general widening of its borders in depth are universally conspicuous (Figs. 8 to 13).

The absence of any doming or tilting of the sediments by the igneous rock is notable in these peripheral sections. Large areas of folded and faulted rocks are gone, and their place is occupied by granite, but the

change has left no record of dynamic disturbance of the antecedent structure. The removal by erosion of the larger part of the original cover of the batholith has lessened the evidence of the manner of this intrusion, but the general trend of that remaining is clear.

The top of the granite—at the high points of the dome—reached the andesite series, which at that time rested upon the beveled edges of the folded sediments. At lower points a varying thickness of the tilted rocks intervened, and in places, notably northwest of Boulder, west of Butte, and southwest of Elliston, fragments of sedimentary rock remain between the granite and the andesite. These beds retain the structure of the early folds, and the sloping beds are truncated by the granite below. Passing toward the flanks of the dome, an increasing thickness of sedimentary rock appears between the granite and andesite, the

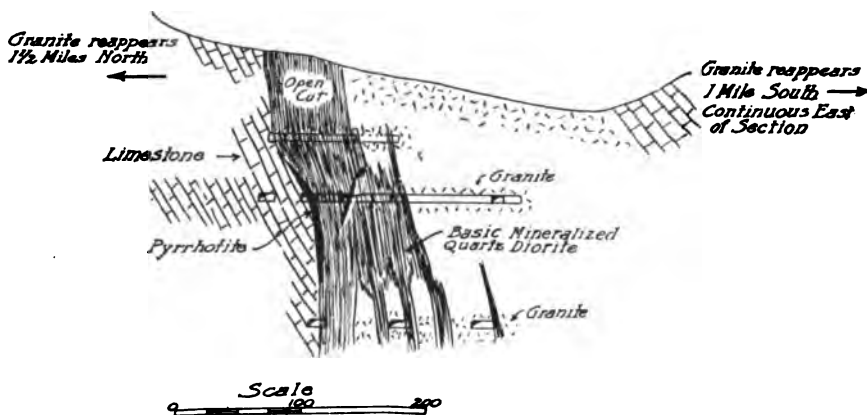


FIG. 9.—SPRING HILL, LEWIS AND CLARK COUNTY. AFTER F. A. LINFORTH.

former showing as irregular intrusions cutting across the bedding independently of structure. It is thus impossible to regard the granite as thrust, in laccolith form, between the andesite and sediments now covered, and it becomes necessary to explain the disappearance of large masses of rock from the area now occupied by the batholith.

The contact effects of the intrusion upon the sedimentary rocks are surprisingly slight. Actual contact phenomena seldom extend more than 1,000 ft. from the igneous rock, and the less susceptible rocks are found unaltered within a very short radius. Recrystallization of the sediments is the most widespread effect of contact action; the addition of iron and silica by magmatic solutions is common; and contact ore deposits of gold-bearing pyrite and chalcopyrite are abundant. In general, pure limestones are recrystallized, and if silica be contributed, tremolite is formed. Magnesian limestones form sandy dolomitic marbles, with development of diopside and chlorite. Siliceous lime-

stones are altered to tremolitic marbles, with contact minerals varying with the impurities of the original rock. Shales form hornstones with or without contact minerals. In quartzites, micas and feldspars are developed, secondary quartz is added, and the original bedding is obscured or obliterated. Andesites show a recrystallization of the ground mass. In all the intruded rocks veins of quartz and specks of pyrite mark contributions from the igneous magma.

The possibility of the fusion or absorption of the sediments into the granite is eliminated by these slight contact effects, as well as by the absence of chemical evidence of such accretions near its borders. The clean-cut truncation of the strata implies mechanical action, the manner of which is suggested by the accompanying sketch (Fig. 14).

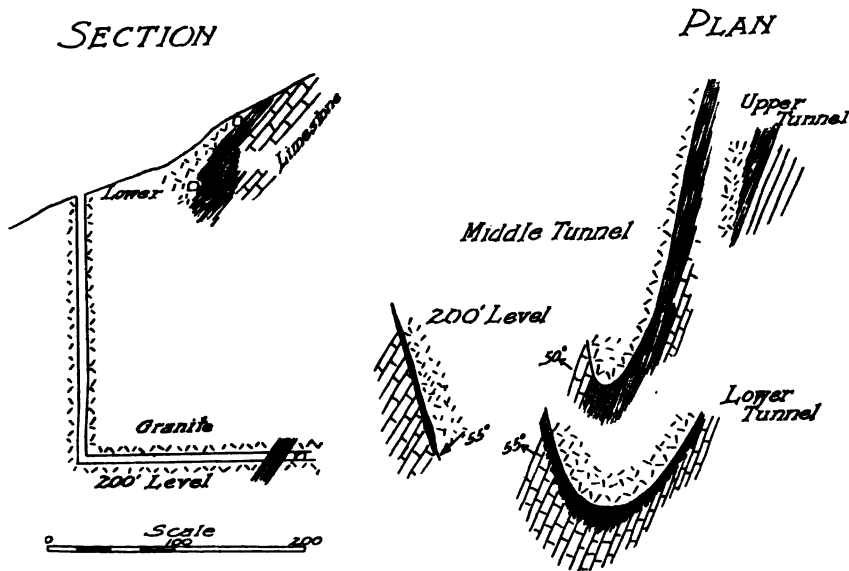


FIG. 10.—COPPER QUEEN, MADISON COUNTY. AFTER RENO H. SALES.

It appears from this that dikes and sills, vanguards of the intrusion, penetrated into joints and cracks of the sedimentary rocks and broke them into detached blocks. Once loosened, these were pushed aside by the further intrusion of the magma, and finally were either caught as inclusions in the solidification of the granite or subsided through the semi-fluid mass to depths below the present exposed surface. Near the contacts inclusions are common for a few score feet into the granite. These are angular, uncorroded, and often of forms that could be pieced together into their original integral mass. Beyond this zone, however, inclusions are rare, and it seems probable that in the slowly cooling magma a majority of the sedimentary blocks were either absorbed or had

ample time to settle far below the depth so far reached by erosion. This explanation conforms with the theory of magmatic stoping which was applied at Marysville by Barrell in 1907. Field observations throughout the area of the Montana batholith have corroborated its soundness.

Evidence of Increasing Acidity in Magmatic Reservoir

The granite itself exhibits three types of contact phenomena. On the southern edge of the batholith the contact is marked by a profusion of basic inclusions, normally under a foot in diameter, but ranging up to 50 ft. These inclusions are in general fine-grained diorite, slightly more basic than the batholith itself. They show a slight alteration on the

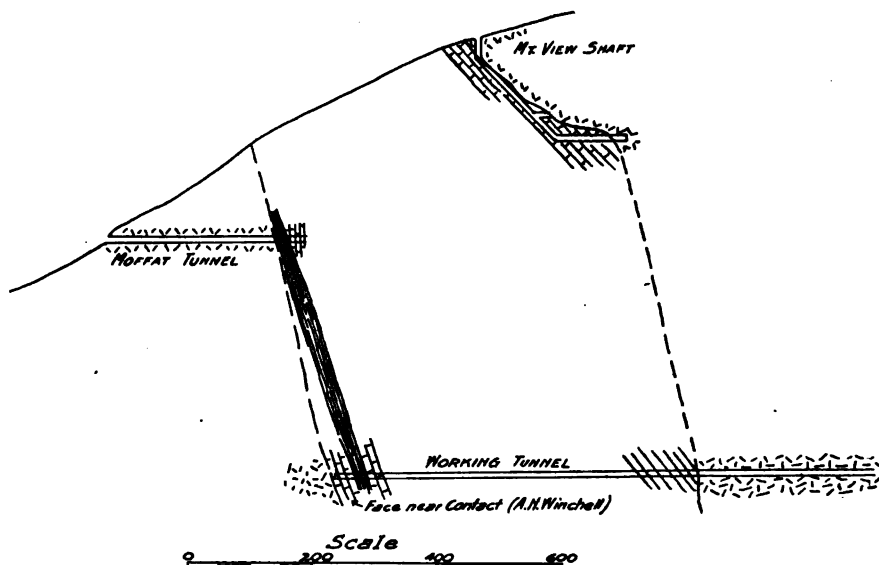


FIG. 11.—BEAR GULCH, MADISON COUNTY. AFTER H. V. WINCHELL.

edges, are generally of angular form, and are distributed within the granite impartially on limestone, quartzite, and andesite contacts. These facts, together with the absence of any phenomena suggestive of magmatic segregation, lead to the conclusion that the diorite fragments represent an early basic crust of the batholith, caught up and included in a further advance of the main magmatic mass.¹¹

On the north, however, the original basic border remains, and in the pyrrhotite and pyroxenite of the Spring Hill mine¹² is found at its extreme development. In this case the distance from the pyrrhotite contact through the gradations to normal granite is about 60 ft., in which

¹¹ Type locality, near lime kiln, Highland Mountains, Silver Bow County.

¹² Report by F. A. Linforth, Anaconda Copper Mining Co., Geological Department, Butte, Mont. (1912).

interval pyroxenite, gabbro, and diorite successively occur. Similar contacts, less perfectly developed, are found throughout the Helena region, and on granite-andesite borders it is frequently impossible to detect the exact point at which the fine-grained basic granite ends and the recrystallized andesite begins.

The third type is of less common occurrence, and is found only where the more acid phases of the granite form the contact. It takes the form of increasing acidity in the outlying dikes and sills of the igneous rock. These, with crystalline continuity, grade from normal granite at the point of departure from the parent mass to alaskite and even pure quartz at their further terminations.

It seems possible to make these contact types conform to the conception of a general increasing acidity of the main magma. The primary contact was a basic segregation, which in a portion of the periphery still

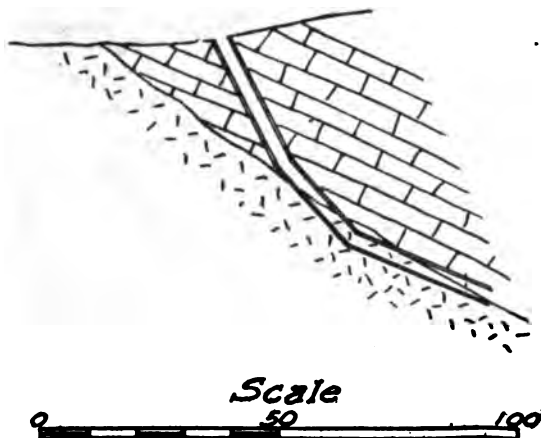


FIG. 12.—IRON MOUNTAIN, ARGENTA, BEAVERHEAD COUNTY. AFTER F. T. GREENE.

exists. Further intrusion on the south pushed the more acid residual magma through and beyond this border, while the more remote penetrations of the igneous rock became increasingly acidic.

The same sequence is evident within the mass of the batholith. The earliest crystallization—apart from the contact—took the form of quartz-monzonite, with about 63 per cent. silica. The composition of this varies but little within the limits of the batholith. While still warm this monzonite was intruded by irregular masses of aplite, which, in varying amounts, is of universal distribution and forms roughly 10 per cent. of the present surface. This contains 75 per cent. silica, and bears witness to the increased acidity of the reservoirs. The third stage is represented by siliceous injections within the aplite itself—quartz and jasper veins and stringers that, with the later addition of sulphide minerals, form the typical ore deposits of the batholith. Pyrite

and chalcopyrite are associated primarily with these siliceous veins, but are not rare as primary minerals in the aplite, and are occasionally found in the monzonite itself. Both are of rare occurrence in the basic contact phases of the batholith, and are in general associated with the siliceous emanations of the later periods.

Ore Deposits of Common Origin

The ore deposits within the granite point to some final conclusions. They are typically fissure veins, enlarged by replacement, generally associated with the aplite, and of nearly uniform east-west strike. Many can be traced for several miles, and widths of 50 or even 100 ft. are not

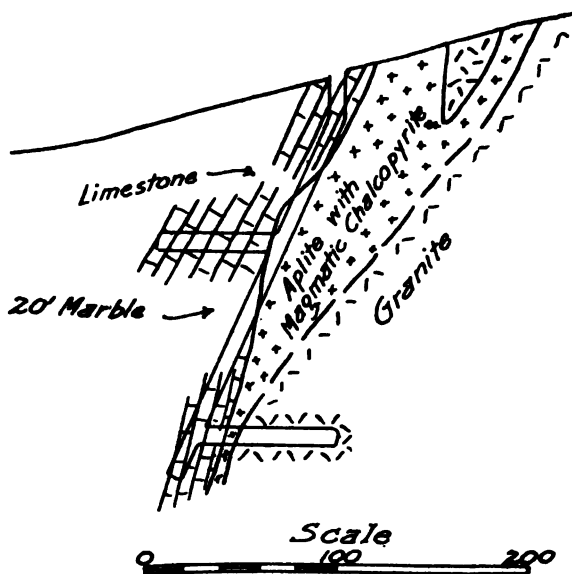


FIG. 13.—MODOC, GRANITE COUNTY. AFTER P. BILLINGSLEY.

uncommon. The widespread and uniform distribution of such veins throughout the granite area bespeaks a common origin, determined by developments within the cooling batholith, and the general association with aplite suggests the ultimate acid residue of the magma as the source of the mineralized solutions.

It is possible to separate the mineral filling of these veins into three periods. The earliest is marked by minerals that indicate the pneumatolytic and semi-igneous conditions then prevailing. Feldspar, vitreous quartz, tourmaline, mica, pyrite, and molybdenite are characteristic. The second period is that of the great sulphide mineralization, with pyrite, chalcopyrite, sphalerite, galena, tetrahedrite, and small quantities of bornite prevailing. This mineralization has afforded the

deposits of the Helena, Wickes, Basin, and (in part) Butte districts. These sulphides are ranged in vertical zones within the veins; galena, with secondary silver enrichment, predominating at the surface; sphalerite most abundant between 400 and 700 ft. in depth; and below only pyrite and sparse chalcopyrite with quartz. This distribution, which conforms to the solubility of the several minerals in cooling solutions, is a universal result of the primary mineralization of the second period. In general all the deposits of this period show all the above sulphide minerals, but it is suggestive of possible variation of the magmatic reservoirs that in the mines north and west of the general line, Butte-Wickes, lead and

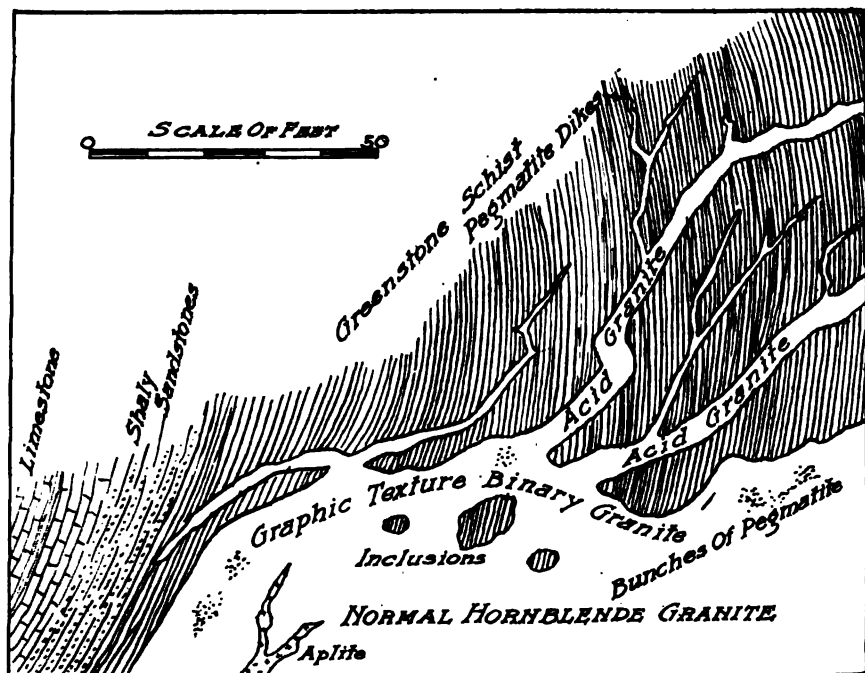


FIG. 14.—DETAIL OF GRANITE CONTACT, LOST CREEK.¹³

zinc, with the secondary silver, predominate, while to the south and east these minerals are in small amounts, with ore shoots of chalcopyrite capped by secondary chalcocite. These orebodies are earlier than the Pliocene dacite-rhyolite, though the numerous dikes may have caused a reconcentration of ore along their flanks.¹⁴

¹³ Camsell (Geology and Ore Deposits, Hedley Mining District) *G. S. C. Memoir No. 2*, 1910, on p., 101 gives the details of a granite contact that affords an interesting parallel to the one shown in Fig. 14.

¹⁴ The prevalence of ore shoots in that portion of the veins adjoining rhyolite dikes suggests this. The Baltimore mine near Boulder, the Comet mine near Basin, and the Hibernia and Nettie mines in Butte are widely separated examples.

The third period of mineralization within the batholith is found only in the Butte district, where a rich copper series—enargite, bornite (rarely chalcopyrite), covellite, and chalcocite—has penetrated from below, and over a limited area replaced the ores of the earlier stage. These border the district, and, with lead and zinc predominating to the northwest and chalcopyrite to the southeast, show the normal vertical distribution of silver, zinc, and pyrite zones. The replacement by copper minerals has proceeded from a center under Anaconda Hill, and has at that point reached the present surface. To the east, north, and west the copper mineralization is found at successively deeper points in the more remote veins, with an increasing cover of earlier ores. This third stage was long continued, the solutions depositing the same sequence of minerals in fissures opened at successive periods. Dikes of quartz-porphry, found only in the Butte district, may bear a genetic relation to this latest stage, but accumulating evidence of the close association of these dikes with aplite masses in depth leaves this point problematical. The copper enrichment may be placed in point of time between the aplite intrusions in the early Eocene and the rhyolite eruptions of the Pliocene.

The evidence of the ore deposits may therefore be summed up as follows: The intrusion of the aplite and general cooling of the batholith was followed by the development of east and west fissure zones rather uniformly distributed. In the aplite areas pneumatolytic vapors were injected into these openings, with great amounts of silica. Mineralized solutions, differing slightly in portions of the batholith, but representing in general the metallic composition of the magmatic reservoirs, occupied and enlarged the fissure zones by replacement of the walls and deposited the sulphide minerals in a common vertical order. In a single instance a further contribution, in the form of rich copper sulphides, was added. In the absence of any known special factors, the localization of this primary enrichment must be attributed to a segregation of copper within the original magma, and its successive concentration by the splitting off of the several intrusions and solutions.

Conclusions

Some general conclusions regarding the Montana batholith may therefore be briefly enumerated:

1. It is of more widespread extent than the original Boulder batholith, reaching westward with insignificant gaps to Wisdom and Philipsburg.
2. It appears in several of the zones of Rocky Mountain structure.
3. Its age is latest Cretaceous or earliest Eocene.
4. It is subsequent to the folding of the sedimentary rocks, and the dips of the beds on its borders are of no significance.

5. It was intruded as a dome-shaped mass, into folded sedimentary rocks capped by andesite. The high points of the dome, near Butte and Basin, penetrated to the andesite, but on the flanks a varying thickness of sediments remains between that and the granite.
6. The intrusion produced only slight metamorphic effects, either by heat or by contributed material, on the intruded rocks.
7. Structure shows that the sedimentary rocks formerly occupying the area now held by the granite have been removed in mass by truncation of the beds, and detailed evidence points to magmatic stoping as the agency involved.
8. Contact phenomena and the internal succession of rock types point to increasing acidity in the magmatic reservoirs.
9. The ore deposits are of a common type, and indicate the same general conditions of origin, with a magmatic source containing more lead and zinc to the northwest and more copper to the southeast.
10. The Butte district requires a special concentration of copper solutions in the local magmatic reservoir.
11. There is no evidence of downward limitation of the granite, or of external origin for the solutions which produced its ore deposits.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Coal-Dust Fired Reverberatory Furnaces of Canadian Copper Co.

BY DAVID H. BROWNE,* NEW YORK, N. Y.

(New York Meeting, February, 1915)

THE use of coal-dust fired reverberatory furnaces, or indeed of reverberatory furnaces of any description, was for the Canadian Copper Co. a matter of necessity, and not of choice. For 20 years smelting had been done in blast furnaces alone, and with the Herreshoff furnaces used prior to 1904 there was no trouble in treating fine ores. But little flue dust was produced, and this, following the time-honored custom, was wetted down and put back with the charge. Whether the flue dust really smelted, or whether it was worn out by being chased around in a circle, was a problem that troubled no one.

With the installation of modern blast furnaces and high-pressure blowing engines, in 1904, flue dust commenced to assert itself. Evidently more dust was made than could be smelted, but so many vital problems engaged our attention at that time that this minor question was pushed to one side.

In 1906, the details of blast-furnace smelting and the conversion of matte had been worked out to a satisfactory conclusion, and the ever-increasing piles of flue dust and fine ore on the stock yard demanded serious consideration. Numerous experiments in sintering, briquetting, mixing with converter slag, forming blocks of flue dust with green-ore fines and cement, and so on, were undertaken. None of these showed much promise. Our problem was still further complicated by the question of treating converter slag. The ore was basic, the slag was not needed as a furnace flux, and it was felt that under these conditions the old method of pouring slag in molds and resmelting in the blast furnace was an unnecessary expense. If the converter slag could be settled in basic-lined reverberatory furnaces, in which at the same time flue dust and green-ore fines could be smelted, two problems might thus be solved.

Reverberatory practice with our ores was, however, an unknown factor. As carried on in the West, on siliceous ores and concentrates, at least 25 per cent. of fuel was required, and even this ratio depended greatly

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upon the skill of the firemen. The lack of skilled labor; the difficulty of recovering unburned coal from the ash by water concentration in our Northern winters, and the difficulty of utilizing it, if recovered, in a plant using no steam power; the uncertainty of the effect of highly basic charges on the hearth and walls, and our entire unfamiliarity with reverberatory practice, caused us to defer our decision.

In the *Engineering and Mining Journal* of Feb. 10, 1906, S. S. Sörensen, describing certain experiments at the Highland Boy smelter, called the attention of the metallurgical world to the possibilities of pulverized coal as a reverberatory fuel. While Mr. Sörensen's experiments did not lead to the adoption of this method of using coal at the Highland Boy, they showed very clearly that increased tonnage, with decreased fuel consumption, could be attained and that such difficulties as he encountered were largely mechanical and presumably removable. Mr. Sörensen was, as far as I know, the pioneer in the use of pulverized coal in reverberatory furnaces.

His experiences were confirmed by Charles Shelby, who in an able article in the *Engineering and Mining Journal* of Mar. 14, 1908, described his experiences in the use of pulverized coal in the reverberatory furnace at Cananea. Mr. Shelby experienced much trouble by the sticking of ash in the flues and the formation of a siliceous blanket over his charge, but until blocked by these conditions he attained better results, both in tonnage and fuel ratio, than had been obtained by grate firing. A profitable contract for the purchase of fuel oil led to the discontinuance of these experiments, but enough had been done to show that the subject was worthy of further investigation.

In October, 1909, I visited Mr. Sörensen at the Steptoe Valley smelter, of which he was then superintendent. We went over the details of the Highland Boy experiments together and agreed that with proper attention to structural and mechanical details the troubles there experienced could be avoided. In the same month I visited Mr. Shelby and discussed the difficulties encountered at Cananea. These also seemed avoidable. It was evident that if the problem could be worked to a successful issue the fuel ratio, then usually about 4:1, might be raised to 6½ or 7:1. This warranted considerable expenditure in working out the details of practice.

In visiting all the prominent Western smelters in that year, 1909, I found that the proposal to use pulverized fuel on a large scale was received with more interest than enthusiasm. As a rule, the profession was skeptical as to the expediency of starting a new plant on a practically unproved method.

During the fall of 1909, our Chief Engineer, George E. Silvester, visited the cement factories in the Eastern States in order to study the proper method of grinding and burning coal. His report confirmed our

opinion, that the process was practical, and during the winter plans were drawn for a reverberatory plant to use pulverized coal as fuel.

The mechanical difficulties encountered at Highland Boy and at Cananea consisted chiefly of two things, viz.: the stoppage of flues with accumulations of ash, and interruptions and irregularities in the coal-dust feed. It had been demonstrated in cement plants that the operations of feeding and burning pulverized coal could be made quite as continuous, as uniform, and as easily regulated as feeding fuel oil, provided only that proper methods were used in the preparation of the coal.

A plant equipped with the latest appliances for drying and pulverizing coal was therefore designed, in a fireproof building entirely separated from the reverberatory-furnace building. Especial care was taken to specify that all bins, conveyors, etc., for the pulverized coal be made as nearly dust proof as possible by the use of rubber gaskets, to eliminate the danger of dust explosions. To circumvent, if possible, the trouble from accumulations of coal ash, an entirely new arrangement of furnace flue was designed, the idea being to eliminate the several right-angled bends in common use, and provide, as far as possible, a straightaway course for the gases. In following out this idea, the skimming door was taken from its traditional position at the end of the furnace and placed on the side, entailing the sacrifice, apparently, of nothing but the tradition.

As the furnishing of steam power from the waste gases was not an essential feature of the installation, hydro-electric power being used exclusively in the plant, the waste-heat boiler was made entirely a secondary consideration, and was situated so as not to interfere in any way with the straightaway idea when not in use.

In February, 1910, Mr. Silvester and I visited the Western smelters to obtain information on reverberatory practice. Mr. Sørensen was keenly interested in Mr. Silvester's plans, in which he advised a few modifications of minor details, while approving the ideas as a whole.

In April, 1910, the Canadian Copper Co. authorized construction, and work was begun at once. As the entire site of the proposed plant had to be raised 11 ft. above the yard level, and a large amount of rock cutting and filling was necessary on the hillside where the bins and approaches were planned, active construction of the reverberatory proper was slow, and operation did not commence until Dec. 23, 1911.

As built, the original furnaces were lined with basic brick, and the hearth was an inverted arch of magnesite. The furnaces went into operation before any proper means of drying flue dust was provided, and during the winter of 1911-12 a large amount of the charge, wet and frozen as it came from the piles, was shoveled in through the doors of the furnace. All the converter slag was poured in, at first through an opening in the roof, later by means of an iron chute through a door near the fire end, as scrap is charged in an open-hearth steel furnace.

The introduction of so much cold air and cold material prevented any satisfactory fuel ratio. During the first five months 21,406 tons of cold charge and 43,463 tons of converter slag were smelted with 9,609 tons of coal. This shows a ratio of 6.7 tons of total charge per ton of coal, but only 2.2 tons of cold charge per ton of coal. However, as the cold charge was wet and often frozen, better results probably could not be expected.

The combustion of fuel was satisfactory from the start, no trouble being experienced either in grinding or burning the coal. The ash, while working on cold charges, choked and clogged the flue at the throat. This difficulty was not eliminated until later, when hot calcines were used and a larger tonnage was smelted. In general, the slower the furnace is worked the colder is the ash and the more it sticks and accumulates, while the faster it is driven the less does the ash hang back in the furnace. Under present conditions, with rapid smelting, the ash is a negligible factor.

In the summer of 1912, the roof and side walls were repaired and some facilities provided for drying the charge. In the winter of 1912, four Wedge furnaces were built to roast green-ore fines. These went into operation in March, 1913. At this date we ceased to send converter slag to the reverberatory furnaces, since with the opening of No. 3 mine the blast-furnace charge became more siliceous and slag could be used economically as a flux.

During this year very pronounced improvements were made by Mr. Agnew, then superintendent of the smelter, who with his foremen, Messrs. Kent, McAskill, and Mason, worked out and adapted to our use a modification of the Cananea system of side fettling. Long and shallow pockets were provided along the side walls, and through holes in the roof green-ore fines were fed to protect the sides. This naturally led to bricking up all the doors on the furnace, and the marked improvement which resulted from the exclusion of cold air and the insulation of the walls by a non-conducting and continuously renewed blanket of fines brought about the extension of this side-fettling system to take in almost the entire charge of calcines.

As the walls were thoroughly protected by the charge thus introduced, the use of basic brick in the walls and hearth was no longer necessary, and the next change, in October, 1913, was to the siliceous bottom and silica brick walls customary in Western smelters.

In 1914, the fuel ratio and furnace practice were steadily improving. The figures for the first three months in 1914, one reverberatory being in use, are given in the tabulation on the opposite page.

In the summer of 1914, a change was made in grinding the ore fines for the Wedge furnace. The ore, which was previously too coarse to make a good calcination, was treated in ball mills, and screened, so that only about 14 per cent. remained on a 20-mesh screen, instead of the former 40 per cent. This finer-crushed ore cannot be produced in sufficient

	January	February	March
Furnace days.....	31	28	31
Calcines, tons.....	10,020	9,460	10,860
Blast-furnace, flue dust, tons.....	906	922	847
Wedge-furnace, flue dust, tons.....	171	193	180
Converter slag.....	69	248	0
Green-ore fines and samples.....	1,731	1,326	2,308
Total charge.....	12,897	12,149	14,195
Coal.....	2,575	2,150	2,094
Charge per day.....	416	434	458
Coal per day.....	83	77	67
Ratio charge to fuel.....	5.0	5.65	6.77

quantity to keep the furnace up to its full capacity. Furthermore, when the calcines dropped, on account of this finer grinding of the ore, from 13 per cent. sulphur to 7 or 8 per cent., the production of slag increased and the production of matte fell off. These conditions, with shortage of calcines, militated against the high ratio of charge to fuel and in June, 1914, the fuel ratio was 5.35.

This historical narrative is introduced to show the gradual development of the process and the conditions which have brought about changes in the original plans.

The following notes on construction are given in explanation of the accompanying drawings, Figs. 1 and 2, which reduced in size, as they must be, are not always clear.

The area occupied by the reverberatory-furnace building was raised about 11 ft. above the level of the surrounding yard, by pouring furnace slag between concrete retaining walls, which were protected as the filling progressed by clay against the concrete. At distances of 56 ft. apart, on the center lines between furnaces, tunnels 12 ft. wide were provided in this slag foundation. These tunnels were to carry tracks so that the reverberatory furnaces built on this poured-slag area could be tapped into pots at the yard level. The furnaces are skimmed into 25-ton pots on this yard level.

Under the lines where the furnace side walls were to go, concrete footings were introduced, and between these footings transverse tie rods were laid in iron pipes and the slag pouring continued. These tie rods carry anchor plates which hold the footings under the furnace walls together and take up the lateral expansion thrust at the foot of the side buckstays. Under the furnace hearth the slag filling rises 12 in. above these concrete footings. On the concrete footings rise the silica-brick furnace walls.

The horizontal area of the furnace is 23 ft. 6 in. by 116 ft. 9 in. outside the brick work.

The side walls rising from the footings inclose 12 in. of poured slag which extends under the silica hearth. The side walls are carried up 27 in. in thickness to a height of 3 ft. $4\frac{1}{2}$ in. and are continued with a thickness of $22\frac{1}{2}$ in. for 5 ft. $4\frac{1}{2}$ in., making the total height of the side walls 8 ft. $9\frac{1}{2}$ in. up to the point where the cast-iron skew-back block is laid for the arch roof. This height is maintained for a distance of 34 ft. from the fire end, from which point the skew backs slope down to correspond to the slope of the arch roof referred to below.

The end or fire wall is 3 ft. 6 in. wide at the bottom for a height of 2 ft. and is then stepped back to $22\frac{1}{2}$ in. at a height of 3 ft. 8 in., and again stepped back to a width of $13\frac{1}{2}$ in. at a height of 6 ft. 3 in. At the other end of the furnace, commonly called the skimming end or front, the construction is very heavy to resist the end thrust of the hearth. It consists of a brick block, 6 ft. wide and 3 ft. high, which is stepped back to a width of 2 ft. 6 in. at the throat, at which point it is 4 ft. 9 in. high.

The roof at the fire end is of 20-in. silica brick. The height at the skew back is 7 ft. $9\frac{1}{2}$ in. above the bottom of the quartz hearth. The central line is 9 ft. $9\frac{1}{2}$ in. above the same point. The radius is 29 ft. $3\frac{1}{2}$ in. on the under side of the arch.

When the hearth is in, the inside arch at the center is 7 ft. $9\frac{1}{2}$ in. to 7 ft. $11\frac{1}{2}$ in. above the top of the hearth and about 6 ft. 8 in. above the skim line, or 4 ft. 8 in. above the center line of the coal-dust nozzles.

This height of arch is maintained for a length of 34 ft. from the outside of the fire wall. In the next 12 ft. the arch drops 1 ft. $10\frac{1}{2}$ in., giving a height of 5 ft. 11 in. to 6 ft. 1 in. above the top of the hearth and about 4 ft. 10 in. above the skim line. This height is continued straight through to the throat of the furnace.

The 20-in. silica arch bricks are used for 34 ft. on the straight arch and for 12 ft. more on the sloping arch. The remaining portion of the roof is of 15-in. brick. As the height of this roof has been changed at various times the heights given for the roof at various points are not exactly as calculated.

There are no side doors on the furnace. As originally built, doors were set at 12-ft. centers, but these have been filled up, so that the side walls present a continuous face of silica brick $22\frac{1}{2}$ in. thick.

The hearth is silica sand tamped into place. No binder was used, though better results might have been obtained had some base been introduced. After about five days' firing 50 tons of high-grade matte was put in to saturate the bottom. If steam from the silica sand came through the walls the heat was cut off for 24 hr. to allow the moisture to escape. Some patches of bottom floated up, but not enough to interfere

with subsequent operations. This bottom is almost flat, being 24 in. thick at the end walls and 22 in. thick at the tap hole, 36 ft. from the fire wall. Another tap hole is provided about 13 ft. from the fire wall.

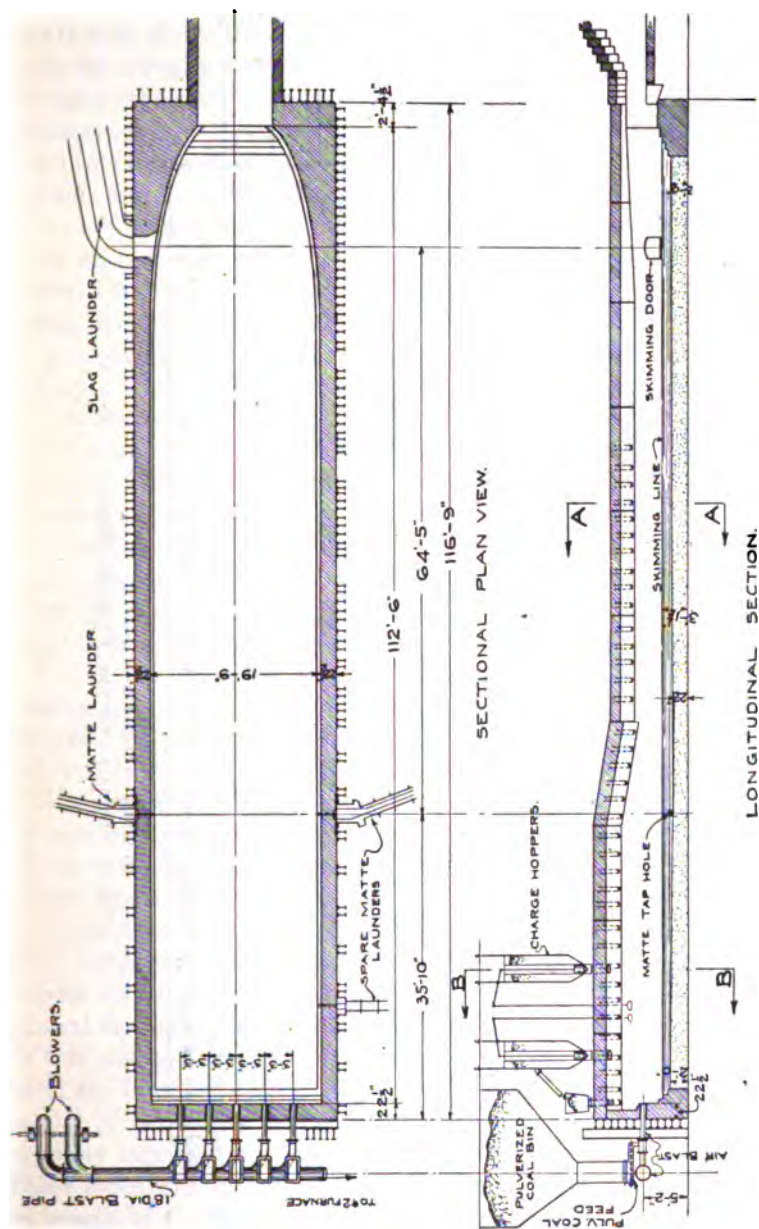
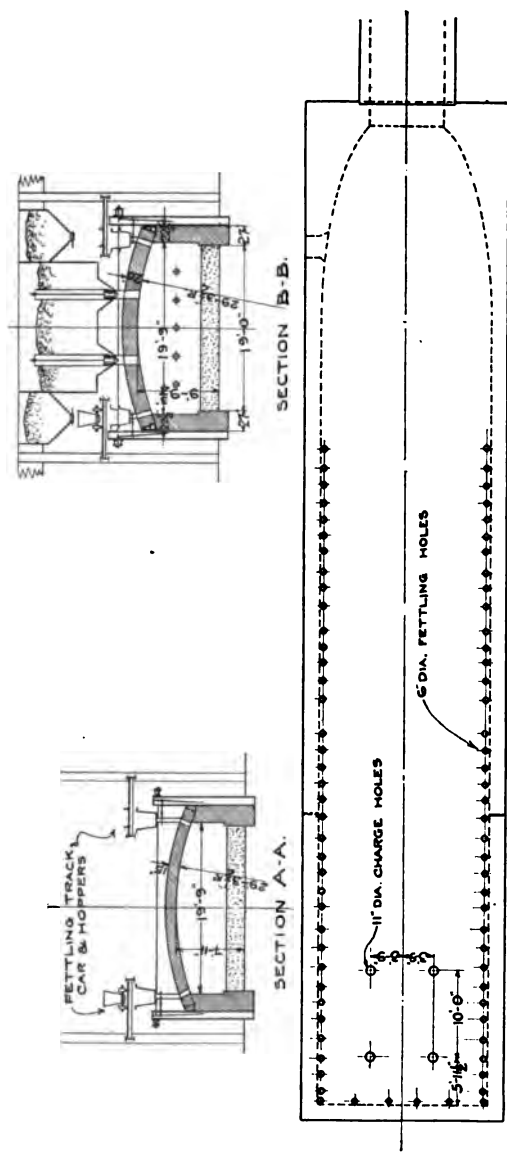


Fig. 1.—No. 1 REVERBERATORY FURNACE OF THE CANADIAN COPPER CO., COPPER CLIFF, ONT., CANADA.

In building the side walls, wood strips are introduced to provide for expansion. These wood strips, $\frac{1}{4}$ in. thick, are placed between every

four bricks on the inside and between every six bricks on the outside. As these burn out they allow the brick to expand horizontally. The arch is laid in separate sections 10 to 12 ft. wide, with the usual wooden expansion wedges 2 or 3 in. thick between the sections.



PLAN VIEW
BINDING REMOVED
FIG. 2.—DETAILS OF THE FURNACE SHOWN IN FIG. 1.

The side walls, built as described, are carried straight to a point 26 ft. from the throat, where they curve inward, the space of 19 ft. 9 in. between

them being narrowed up along a line of gradually increasing curvature to a width of 8 ft. 8 in. at the throat. At this point the opening is 4 ft. 3 in. high at the center and 3 ft. 9 in. high at the sides. The arch here is about 4 ft. 8 in. above the skimming line.

From the throat a straight flue 8 ft. 8 in. wide leads to the waste-heat boilers and to the stack. Openings are provided along the side of this flue for cleaning out any deposited ash. An opening opposite to the throat is provided by raising the bottom of this flue about 18 in. above the throat and introducing a door in the space thus formed. This is very useful for removing any accretions of ash fused in the throat. The skimming door is placed on one side of the furnace, 16 ft. 6 in. back from the throat. This door, 2 ft. 6 in. wide by 15 in. high, allows slag to run off down to a skimming line $14\frac{1}{2}$ in. above the hearth at the tap hole. The slag can rise 6 in. above this line before reaching the level of the side doors now bricked up. Outside the skimming door a cast-iron clay-lined box is provided to trap any matte carried over. From this a cast-iron slag launder curves to a line almost parallel with the furnace and delivers the slag into 25-ton pots, which are brought in on a track at right angles to the furnace under the flue.

The furnace is fed in a rather peculiar way. When the furnace was started almost all the charge was introduced through two charge hoppers near the fire end, as in the usual Western practice. The first hopper delivered through two openings 11 in. in diameter, 7 ft. 6 in. apart and 8 ft. from the outside of the fire wall. The second hopper delivered through two similar openings 18 ft. from the fire wall.

At present almost all the charge is introduced through hoppers along the side walls. Directly over the side walls, at the fire end of the furnace, large bins are provided, which discharge into small bottom-dump cars. These cars run on 24-in. tracks which are supported from overhead. Under these tracks a long trough extends down each side of the furnace just above the side walls. These troughs are filled from the cars on the track above them. Each trough has openings in the bottom, 2 ft. apart, which openings communicate by a slide gate with 6-in. iron pipes. These pipes pass into holes drilled in the roof bricks which allow the charge introduced through these openings to slide down on the side walls, over which this charge forms an almost continuous blanket. As there are no doors on the furnace, and as the 6-in. pipes are clayed into the openings in the roof, it follows that no air is introduced into the furnace except what is purposely introduced at the fire end.

These pipes form a continuous line of charging holes, which extend the entire length of the furnace. The charge on the side opposite the slag door is fed all the way to the throat. On the slag side it is fed along as far as the slag door and no farther, as the cold air coming in while skimming cools the wall from the skim door to the throat and obviates the necessity

of charging beyond this point. Six similar openings are used on the fire wall.

The walls are held in place by 12-in. I-beams in pairs, with a space of 5 ft. between each pair, which form the side braces. These are wedged in at the bottom by wooden wedges against an iron strap in the concrete footings. The concrete footings are tied together as previously described by rods passing under the furnace. At the upper end the 12-in. I-beams are tied across the furnace by $1\frac{1}{2}$ in. rods.

The coal dust is introduced through five pipes, 5 in. in diameter. One of these pipes is on the center line of the furnace; the others are in horizontal line with it at distances of 3 ft. 3 in. from center to center. These pipes are 5 ft. 2 in. above the bottom of the sand hearth, or 3 ft. 2 in. above the top of this hearth. They are about 2 ft. above the skimming line of the charge and the central pipe is about 4 ft. 8 in. below the highest point of the roof.

The coal used in firing is a good quality of slack. Analysis of one lot showed: Volatile matter, 34.70; fixed carbon, 55.40; ash, 9.45; sulphur, 1.30; moisture, 4.31 per cent.

This coal has a thermal value of about 13,500 B.t.u. per pound. It is about $\frac{3}{4}$ in. and under in size and contains about 7 per cent. moisture. It is dried in a Ruggles-Coles drier, 70 in. in diameter and 35 ft. long. One ton of coal burned on the grate dries 40 to 50 tons of slack coal to about 5 per cent. moisture, which falls to 2.4 per cent. moisture after grinding. About 10 tons of slack is dried per hour of running time. The coal is ground in Raymond impact mills. About 95 per cent. passes a 100-mesh and 80 per cent. passes a 200-mesh screen.

The pulverized coal is sucked by a fan to separators above the roof of the drier building, and slides down into a screw conveyor which delivers it into bins at the fire end of the reverberatory. The dust is fed from these bins by Sturtevant automatic-feed screw conveyors, one for each nozzle, the speed of which can be regulated. These screws carry the dust forward and drop it into the air nozzles about 2 ft. from the point where the nozzles enter the furnace. Any coal-delivery pipe can be closed off by a slide gate, and any screw conveyor can be stopped by disconnecting the bevel gears attached thereto. In this way any desired number of the five burners can be run, and at any desired speed within wide limits. The amount of air delivered to each nozzle can be varied at will or cut off entirely.

As a rule the five burners are in operation. Each delivers about 13.5 tons of coal dust a day or about 19 lb. of coal per minute to the furnace. The total coal blown in is about 67 tons a day.

The dust drops from the conveyors into the air pipes which carry it forward into the furnace. The air is supplied by a 4-ft. Sturtevant fan, running at about 1,300 to 1,400 rev. per minute. The air supplied by this

fan is insufficient for the combustion of the fuel. Openings are left in the end wall between the coal burners. These openings are stopped by loose bricks, so that the amount of air is readily controlled. The draft at the fire wall is about 0.25 in. of water and at the throat it has a maximum of about 1.2 in. The combustion is very good. One test made for 10 days (Jan. 9 to 19, 1914) showed the following averages:

Coal consumption, tons in 24 hr.....	69.7
Gas temperature at throat, degrees centigrade.....	922
SO ₂ and CO ₂ , per cent.....	12.3
Oxygen, per cent.....	6.5
SO ₂ , per cent.....	1.14

During this test the average charge was 409 tons in 24 hr. This shows a ratio of 5.9 parts charge to 1 part coal, but much higher ratios have been attained. The average for March, 1914, was 6.84. This coal ratio depends largely upon the composition of the charge and the analysis of the slag produced.

A criticism might be made of the low temperature of the gasses at the throat, 922° C. The usual practise in Western smelters is to carry a temperature of 1,200° to 1,300° at this point, and it might be thought that this low temperature indicated inefficient firing. The fact is that the heat of combustion is utilized in smelting ore along the side walls and consequently the escaping gases, having done more work than is usually the case, are relatively cold. The function of a reverberatory furnace is to melt ore, and not to raise steam, and for this reason the more heat that is absorbed from the coal gases in the furnace, the more efficient the operation and the cooler the escaping gases.

The ash from this coal causes very little trouble in operating. A small amount settles on the slag, but as this slag is high in iron it is not an undesirable feature. A small amount also settles in the flue and a few hundred pounds may stick around the throat. The ash, where exposed to high heat, forms a very light pumice-like fragile mass. The throat is cleaned out daily by opening the door under the flue. During this cleaning the firing is maintained as usual.

The great advantage of coal-dust firing is the absence of the usual breaks in the temperature curve due to grating or cleaning the hearth, and as a consequence a greatly increased tonnage and fuel ratio. The operation of firing, being purely mechanical, comes under the immediate and direct control of the furnace foreman and responds instantly to his regulation. In addition to this, the peculiar method of feeding by almost continuous side charging obviates the breaks in the temperature curve due to charging or ordinary fettling. For these two reasons the chart of the temperature shows an almost horizontal line, rising or falling in almost exact concordance with the speed of the coal-feeding device.

The maximum bath of matte and slag is 22 in. deep. A constant bath of 8 in. of matte is carried. This matte lies 6 in. below the skimming plate, so that after skimming there are 6 in. of slag and 8 in. of matte left in the furnace, making a total minimum depth of 14 in. The skimming door is banked up 8 in. with sand, so that just before skimming the slag is 14 in. deep. As the charge along the side walls occupies a great deal of room there is never at any time more than 40 or 50 tons of slag in the furnace.

In rebuilding this reverberatory, or in designing a new plant, the hearth should be widened to provide for a larger body of matte, which experience has shown to be necessary. As this method of burning coal and of admitting the charge into the furnace bids fair to come into general use it is to be expected that many changes, both in construction and operation, will be introduced. My belief is that reverberatory smelting along these lines will become cheaper than blast-furnace smelting and that a wider range of ores can be used in such a furnace than in the old-style coal or oil-fuel furnaces.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Safeguarding the Use of Mining Machinery

BY FRANK H. KNEELAND, NEW YORK, N. Y.

(New York Meeting, February, 1915)

SAFETY FIRST is a popular motto—most mining companies have adopted it. It is probable, however, that in the majority of cases it is only a motto and gets no further than the office stationery or the bulletin board at the mine's entrance.

In but few industries is there employed a greater diversity of machinery and mechanical devices than in coal mining. The list ranges all the way from mechanical stokers to wood-working machinery and undercutters. Many of these devices, particularly those employed in wood working, are extremely dangerous.

In 1913, 23 per cent. of all fatalities occurring in coal mining in the U. S. were caused by machinery of some sort or other. Although the forms of accidents occurring with mining machinery are legion, they generally arise from one or more of five causes: (a) Falls from ladders, platforms, etc.; (b) coming in contact with moving machines or parts thereof; (c) electric shocks; (d) failure of some machine part; (e) mismanipulation of valves, levers, switches or other hand-operated controlling devices.

The steps which may be taken to prevent accidents from the above causes are almost as numerous as the accidents themselves. There is, and can be, no panacea for all mishaps, nor for any one class of accidents. There is no mathematical or other formula that will bring immunity under any set of conditions. Common sense is the only guide.

The remedy for the first-named class of accidents is simple and generally effective—namely, make the staging, ladder, platform, or other support, abundantly strong to carry any possible weight that it may be called upon to bear; provide ample railings on platforms or runways, and non-slipping feet on movable ladders.

Coming in contact with moving machine parts is one of the greatest dangers met with in the operation and maintenance of mining machinery. Unfortunately, machine manufacturers have not as a class adopted the idea of thoroughly protecting and incasing the dangerous parts of their product. No one has, or should have, a better or more thorough knowledge of what the dangerous parts or elements of a machine may be than its builder. The desire to construct a piece of mechanism as cheaply as

possible impels many manufacturers to forego making the machines as safe as possible.

An exposed train of gears, an unprotected revolving bolt head, set-screw, feather or spline, an open keyway on a shaft, and the like, are all well-recognized dangers. Yet few indeed are the machines where these are all absent. The means to be employed to prevent accidents from these causes are two in number: First, eliminate so far as possible angular projections on revolving or moving parts; *e.g.*, replace ordinary square-head setscrews with those of the hollow variety which screw down flush. Second, guard or protect the moving parts of the machine.

When purchasing new equipment many firms specify something like the following: All angular moving parts or projections must be avoided as far as possible. If impossible to avoid their use, all gears, cams, eccentrics, setscrews, bolts, sprockets, chains, belts, feathers, splines, keyways, cranks, connecting rods, or other dangerous revolving or moving parts actuated by power, must be so guarded and protected as to render contact, intentional or otherwise, between them and the anatomy or clothing of the attendant as nearly impossible as the work to be performed will permit. All guards must be so constructed and built up as to be readily removable without the use of special tools, allowing ready access to the working parts for inspection or repair. It must be the aim of the builder to produce a machine that shall be as safe to handle and operate as human ingenuity can devise and the work in hand will permit, and no bid will be considered unless detailed plans are submitted which shall meet the entire approval of our engineers and safety experts.

But many machines are at present in use which were not purchased on the above or similar specifications and which represent too great an investment to be discarded. To render these machines even reasonably safe, adequate guards are necessary. These guards may be of many descriptions, but in general their object is to render an accident impossible. This is, however, out of the question, since the human element is an unknown quantity, and while it may be possible to make a machine fool-proof, it is difficult to render it impregnable against suicidal intentions.

Several different materials may be employed with fairly good success. Wooden fences or boxes are in general the cheapest in first cost, but are obviously open to many objections. Hand railings and the like made of small structural shapes are, generally speaking, equally efficient and far more permanent than their wooden counterparts. Wire mesh of suitable weight, properly stiffened with structural-shape frames, so constructed as to be easily separated into sections, is one of the most efficient and permanent of all guards. Recently, expanded metal has been introduced in lieu of the wire mesh and appears to be admirably adapted to guarding purposes.

Generally speaking, an air gap of sufficient width is a certain protection

against dangerous electric shocks. While insulation gives supposed immunity, the only safe method of procedure is to treat every electrical conductor, regardless of covering or supposed voltage, with the same respect that would be given were it known to carry a large amperage at high potential. Although underground circuits at American mines usually carry direct current of a voltage not exceeding 500, yet we not infrequently hear of men being either injured or killed through forming a short circuit between trolley and rail. Trolley wires may be protected from accidental contact by comparatively simple means. The trolley guard board, and its means of suspension, are well known and need no further comment. Such protection should assuredly be placed at points of danger, particularly where men pass under the wire, and at other places where observation would indicate that accidents might occur.

A point of danger which exists at many mine plants, and one which is perhaps not generally recognized, is the space behind switchboards, particularly those carrying high-tension conductors or making high-tension connections. Such places, regardless of whether the current carried is direct or alternating, or whether the voltage is 110 or 110,000, should be effectually fenced in, and no one should be allowed behind the switchboard while current is on.

The failure of machine parts is the least probable of all accidents, but one of the most difficult to forestall. It is practically impossible to determine the existence of a faulty weld in a steam pipe, or a cold shut in a casing. Both may withstand a test pressure yet fail in service through the fatigue of materials.

It is well in designing a plant to provide good means of egress from the vicinity of all pipes carrying a high pressure of steam, air, or hot water since the greatest danger from their failure is not so much from flying pieces as from the fluid released. In certain instances, breaking machine parts, dangerous in themselves, may be rendered harmless by proper means. The best known of such devices, although not the only ones, are gauge-glass protectors on boilers and safety collars on emery wheels.

Mismanipulation of hand-operated controlling devices is a prolific cause of accident, of which the overwind is perhaps the best known example in mining experience. Another type of apparatus which causes many deaths annually is the electric switch, which if thrown while a man is making repairs to the line it controls introduces an extreme hazard to the repairman. Boiler washers while at work are sometimes killed through the opening of some valve connecting to another boiler or to the feed pump.

There are upon the market many types of reliable speed regulators, automatic stops, and devices to prevent overwinding which may be attached to any hoisting engine, either steam or electric. A switch controlling a circuit should be provided with means for locking it open, and

the person making the repairs should see that the switch is so secured and have the key in his pocket. In the case of the ordinary single or double knife switch, a hole properly placed in the switch blade an a paddlock answer every purpose. So far as valves are concerned, the wheels may either be chained and locked, or a cover placed over the valve wheel, which when padlocked in place renders it impossible to move the hand wheel.

The above are a few of the preventive means of forestalling accident; the list might be greatly extended.

It is an old and a true adage that "A word to the wise is sufficient." Placards and danger signs, if properly distributed, and, what is vastly more important, if properly heeded, are invaluable. Such signs, however, should be of a character to be universally understood and therefore should be, so far as possible, pictorial or symbolic; that is to say, wordless. The clenched hand grasping the thunder bolts comes as near to conveying a definite idea to all who see it as does the wooden Indian in front of a cigar store. The red circle or disk has its significance also and needs no explanation.

The mental attitude of the employer and his lieutenants is a powerful factor in accident prevention. The superintendent, the foreman, and the pit boss are logical leaders to whom subordinates naturally, although perhaps unconsciously, look for example. If they are reckless, recklessness is apt to prevail among the men. If they are eternally cautious and vigilant, caution and vigilance will be inculcated. It does little or no actual good for a shop superintendent or foreman, for instance, to tack up a danger sign beside a motor and then send a man up a ladder to oil a running lineshaft upon which there are projecting setscrews, or keys, or boltheads. The foreman's example is vastly more powerful than his admonition.

In no branch of mine work should discipline be more strict, than in the enforcement of safety regulations. Breaches of ordinary rules mean little more than slight added expense to the mine management. Disobedience of safety precautions endangers not only the mine, but every living creature therein. The distinction between the two should be as clearly defined as that between civil and criminal actions in law.

The attitude of the mine workers throughout the country varies widely. In Arkansas some time ago the miners at a plant went on strike because certain safety statutes were not complied with. The employees of one of the largest anthracite companies of Pennsylvania strongly protested against, and even threatened to strike because of, the inauguration of what they called the "patrol" system, wherein a sub-foreman visits each working place at frequent intervals to see that all is safe and in accordance with the rules of the company. In many instances, the miner and mine laborer at least have failed to grasp thoroughly the idea that safety regulations are made and enforced for their

sole benefit, and that the necessity for the sub-foreman's instruction to set a prop or take down a dangerous piece of slate is a direct reflection upon their ability as efficient and careful miners. The good and prudent coal digger never fears the visit of the sub-foreman any more than the honest banker dreads the periodic call of the State bank inspector.

Among men who handle mining machinery there is far less antipathy to safety rules and regulations. The mental attitude of these men as a class is much more favorable to greater caution and security than that of the miners. The result is that they are much less prone to object to even decidedly stringent measures and frequently become safety enthusiasts.

Much effort, time, and money have been spent throughout the country during the past few years in first-aid work. This, although by no means a preventive measure, has perhaps exerted a far more powerful mental influence than all others put together. The first-aid field day, the working out of problems involving one or more imaginary, yet possible, injuries, the continual binding up of dummy fractures, etc., cannot fail to make a decided impression upon all who see them, particularly the participants. What we need in mine safety is prevention, not cure, and first-aid is not primarily preventive.

Superstitions and deep-rooted ideas are alike tenacious and require both time and effort for their effacement. We have long been accustomed to regard the calling of the coal miner as being inherently dangerous and requiring a certain amount of recklessness and bravado. The man who would not, on occasion at least, take a chance was not a good miner. And this idea has prevailed since mining began.

The movement for greater security to life and limb, therefore, is one which has to combat, not only existing physical conditions, but all the precedent, precept, and prejudice of the past. It is no wonder, then, that progress is slow, and that it often appears, even to the most sanguine, that a point has or soon will be reached beyond which material headway cannot reasonably be made. This is, however, only a feeling common to all movements toward betterment. Many of us can well remember when an 80-ton locomotive was considered as being about the largest that would ever be commercially successful. Just as the difficulties in the way of heavier rolling stock on the railroads lay not so much in the materials employed as in the preconceived ideas of the builders, so the barriers to greater safety in mining lie in the mental attitude of those concerned rather than in existing conditions.

It is futile to cover a machine with guards and safety appliances and depend upon them alone for protection. These are in the true sense effective only when supported and reinforced by discipline and morale. No guard ever devised is adequate to protect the man who is always careless. Habit is stronger than nature, and in mining no man is safe unless he is safe habitually.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Experiments on the Flow of Sand and Water through Spigots

BY R. H. RICHARDS, BOSTON, MASS., AND BOYD DUDLEY, JR., STATE COLLEGE, PA.

(New York Meeting, February, 1915)

IN nearly all ore-dressing operations it is a common practice to discharge mixtures of fine ore and water through spigots; for example, from classifier pockets, from jig hutches, from settling tanks, etc. As a general rule it is desirable to regulate the composition, *i.e.*, the ratio of water to sand, of such discharged products; and this can usually be done by varying the size of the orifice or spigot through which the material is discharged. Thus it is that problems similar to the following are frequently encountered: With a given quantity of solids per unit time, of a given specific gravity, to be discharged with water under a given head, and with a given ratio of water to solid, what must be the size of the spigot opening? If the spigot is to be open continuously, there are, aside from its form, but three factors governing the rate at which it will discharge a mixture of sand and water, namely: the head of water above the spigot; the area of the spigot opening; and the viscosity of the material to be discharged. The term viscosity, as used here, means the ratio of the volume of pure water that will flow through a given orifice under a given head in a given time to the volume of the material under consideration that will flow through the same orifice in the same time under the same conditions. While the actual viscosity of water containing sand in suspension is not increased by the presence of the sand, nevertheless the volume-rate at which such a mixture will flow through an orifice is less than that at which pure water will flow, by reason of the friction of the particles of sand against each other and against the sides of the orifice. In other words, the flow is retarded as it would be by increased viscosity; and therefore, for the purpose in view, we may speak of the viscosity of a mixture of ore grains and water as defined above. The head of water above the spigot is generally fixed by the form and design of the classifier or tank to which the spigot is attached. So, with the head known, in order to determine the size of the spigot required it is necessary to know the viscosity of the desired mixture, and the coefficient of discharge of

the form of spigot employed when pure water is the material being discharged. From these data, the area of the spigot opening can be calculated from the equation,

$$a = \frac{fq}{c\sqrt{2gh}}$$

in which a is the area of the spigot opening; f , being the viscosity of the mixture, q , the rate of discharge by volume; c , the coefficient of discharge; g , the acceleration due to gravity; and h , the head of water above the spigot.

The coefficient of discharge is the ratio of the actual rate of discharge of pure water to the theoretical rate of discharge as calculated from

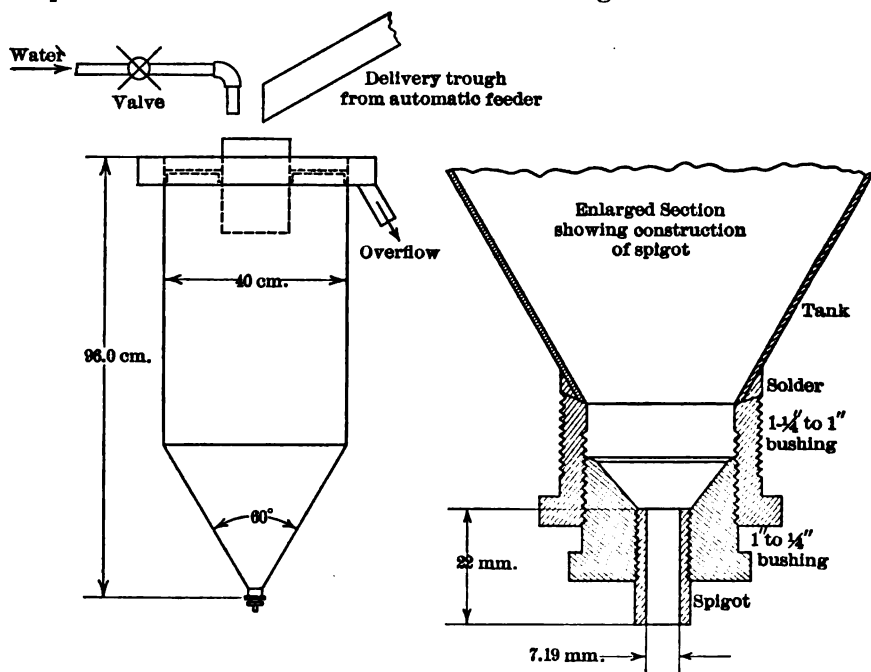


FIG. 1—APPARATUS FOR TESTS OF FLOW.

the effective head above the orifice or spigot. The value of c depends upon the form of the orifice or spigot; some representative values taken from Merriam's *Treatise on Hydraulics* (8th ed.) are given below. For orifices having a sharp inner edge, which is alone touched by the water, $c = 0.59$ to 0.63 , being greater for low than for high heads, greater for rectangles than for squares, and greater for squares than for circles. For a standard short tube (a cylinder having a length equal to about three times the diameter) not projecting into the space occupied by the water, $c = 0.79$ to 0.83 , being greater for low heads than for high heads. For a standard short tube projecting into the space occupied by

the water, $c = 0.72$. For short tubes not projecting into the space occupied by the water and having a cone- or bell-shaped mouth on the influx end, $c = 0.85$ to 0.95 . The latter form is the one usually found in classifier spigots, and a number of tests on the spigot shown in Fig. 1 under heads of from 1 to 3 ft. gave values for c between 0.87 and 0.91 —the higher values for the lower heads.

The experiments described below were performed with the purpose of securing data as to the relation between the composition and the viscosity of mixtures of sand and water. The work was done during the winter of 1911-12 in the ore-dressing laboratory of the Massachusetts Institute of Technology.

The apparatus used in the tests consisted of a cylindrical tank of galvanized iron, having a conical bottom to which the spigot was attached, and a mechanical feeder for delivering dry sand to the tank. The details of the construction of the tank and spigot and of the arrangement of the apparatus are shown in Fig. 1. The tank was mounted with the top or overflow edge level; and, by feeding enough water into it to supply the spigot and to maintain a slight overflow, a constant head of water was maintained above the spigot. Below the tank a swinging sheet-iron launder furnished the means of deflecting the spigot discharge into a bucket or other receptacle for any desired time, and thus determining the rate of flow.

The method used in conducting a test was as follows: The tank was filled with water and the rate of inflow so adjusted as to supply the spigot and to maintain a slight overflow. The feeder was then started and sand was delivered to the tank. The sand settled through the water and passed out of the spigot. The amount of overflow was thereby increased, not only on account of the added volume of sand, but also because of the decreased rate of discharge due to the passage of the sand through the spigot. Nevertheless, this increase in the amount of overflow did not produce a measurable increase in the head above the spigot, because of the great length of the overflow edge and because the actual increase in the rate of overflow was comparatively slight. When the sand and water mixture had been running from the spigot for a minute or so and the flow had become steady, the spigot discharge was deflected into the bucket for a measured length of time, usually about 2 min. The feeder was then stopped, adjusted to deliver sand at a different rate, and another test was made. The discharged product obtained from each test was weighed to determine the total amount of water and sand. The water was then decanted from the sand and the latter was dried on a steam table, after which it was weighed.

The sand used in all the tests was prepared by crushing Roxbury pudding-stone (a siliceous conglomerate) with rolls to pass a wire-cloth screen with 1.4-mm. square holes, and removing the sand and slime finer

than about 0.1 mm. by hydraulic classification. The specific gravity of the prepared sand was 2.72.

The results of the experiment are shown in the following table. The rates of flow are all expressed in kilograms and liters per minute.

Table I.—*Relation of Composition to Viscosity of Mixtures of Sand and Water*

Kilograms Sand and Water	Kilograms Sand	Kilograms and Liters Water	Liters Sand	Liters Sand and Water	Per Cent. Sand by Volume	Per Cent. Sand by Weight	Viscosity of Mixture
9.20	0.00	9.20	0.000	9.20	0.00	0.00	1.00
9.30	0.45	8.85	0.165	9.02	1.83	4.84	1.02
9.35	1.10	8.25	0.405	8.66	4.68	11.8	1.06
9.35	1.40	7.95	0.515	8.47	6.08	15.0	1.09
9.40	1.90	7.50	0.699	8.20	8.53	20.2	1.12
9.40	1.95	7.45	0.717	8.17	8.78	20.8	1.13
9.55	2.20	7.35	0.809	8.16	9.92	22.0	1.13
9.20	2.25	6.95	0.827	7.78	10.6	24.4	1.18
9.05	2.50	6.55	0.920	7.47	12.3	27.6	1.23

The relationship between the composition and viscosity of the mixtures is shown graphically in Fig. 2. The ordinates of the upper curve are the percentages of the sand by weight, while those of the lower curve are the percentages of sand by volume. The viscosities of the mixtures are plotted as abscissas in each case.

It was found in these tests that when the amount of sand in the mixture exceeded 30 per cent. by weight the spigot would produce a very thick discharge for a short time but that its continuous operation under these conditions was not certain; clogging resulted sooner or later. It is therefore concluded that, with conditions similar to those under which these tests were made, a mixture of sand and water cannot be successfully discharged from a spigot when the mixture contains more than about 30 per cent. of sand by weight. In the case of the sand used in these experiments, this figure corresponds to about 13 per cent. of sand by volume, and it should be noted that it is the volume-percentage, rather than the weight-percentage, which governs the viscosity. With sand of greater density, a mixture having more than 30 per cent. of sand by weight could of course be discharged; but it seems doubtful whether a mixture containing more than 13 per cent. of sand by volume would pass continuously from this spigot if the sand grains were 1.4 mm. in diameter. On the other hand, with the same size of grain and a larger spigot or with smaller grains and the same spigot a more concentrated mixture could doubtless be discharged.

There is much more experimental work to be done along this line,

before a complete and satisfactory set of data will be available. Two points that call for investigation are the effect of grain size and the effect of the density of the grains upon the viscosity of the mixtures; another is the relationship between the ratio of the spigot diameter to that of the maximum grain and the mixture of maximum concentration that will flow.

A concrete example, illustrating the use of the data given above, may prove of interest. It is desired to discharge from the pocket of a classifier 40 tons of sand per 24 hr. together with water in the ratio of 1 part of sand to 3 parts of water, by weight. The head of water above the spigot

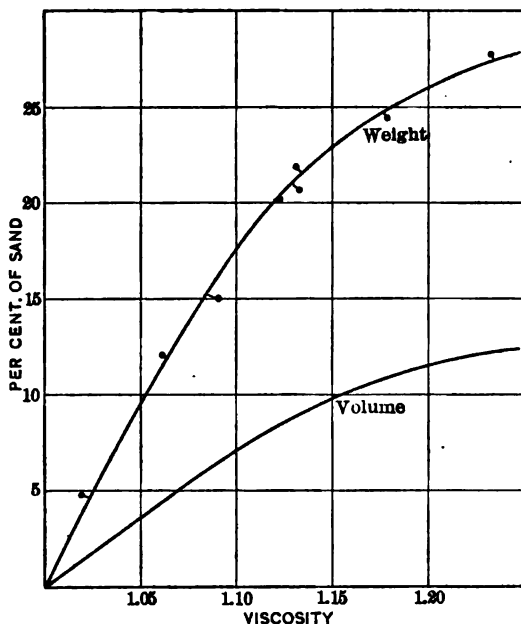


FIG. 2.—GRAPHIC REPRESENTATION OF RESULTS SHOWN IN TABLE I.

is 3 ft. The form of the spigot is that of a short tube with a cone mouth on the influx end. The mean specific gravity of the sand is 2.81. What must be the diameter of the spigot opening? For the sake of convenience metric units are used in making the calculation.

The area of the spigot opening may be obtained from the formula given above,

$$a = \frac{fq}{c\sqrt{2gh}}$$

Taking up the terms on the right hand of the equation in order, f , the viscosity, may be estimated as follows: The weight-ratio of water to sand in the mixture to be discharged is 3 to 1. Considering 100 g. of the mixture, the weight of water is 75 g.; its volume is 75 cc. The volume of the sand is 25 g. $\div$ 2.81 (the density of the sand) = 8.9 cc. The total

volume of 100 g. of the mixture is $75 + 8.9 = 83.9$ cc. Hence, the percentage of sand by volume in the mixture is $8.9 \div 83.9 = 10.6$. From the lower curve of Fig. 2, the viscosity of a mixture containing 10.6 per cent. of sand by volume is 1.17. Therefore $f = 1.17$.

The quantity of sand discharged per 24 hr. is 40 tons. One ton per 24 hr. is 0.631 kg. per minute. Forty tons per 24 hr. is $40 \times 0.631 = 25.2$ kg. per minute. The volume of sand per minute is $25.2 \div 2.81$ (the density) = 8.98 liters. The quantity of water per minute is three times that of the sand, $25.2 \times 3 = 75.6$ kg. = 75.6 liters. The total volume of sand and water per minute is 8.98 (sand) plus 76.5 (water) = 85.5 liters. The total volume per second is $85.5 \div 60 = 1.43$ liters = 1,430 cc.

Since the spigot is to consist of a short tube with a cone mouth on the influx end, the coefficient of discharge, c , may be assumed as 0.88.

In metric units $g = 980$ cm. per sec.²; and the head, h (3 ft.), is 914 cm.

Substituting these values in the above equation gives, for the area of the spigot opening

$$a = \frac{1.17 \times 1430}{0.88 \sqrt{2 \times 980 \times 914}} = 1.42 \text{ sq. cm.}$$

The diameter may be obtained from the relation,

$$d = 2\sqrt{\frac{a}{\pi}}, \quad d = 1.35 \text{ cm.} = 0.53 \text{ in.}$$

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Coal-Dust Fired Reverberatories at Washoe Reduction Works

BY LOUIS V. BENDER, ANACONDA, MONT.

(New York Meeting, February, 1915)

AFTER investigating the work of coal-dust fired reverberatories of the Canadian Copper Co., at Copper Cliff, Ontario, the management of the Washoe Reduction Works decided to experiment with and ascertain the advantages of using coal dust as a fuel in their reverberatories. Consequently, during the month of June, 1914, reverberatory furnace No. 8 was changed in order to use powdered coal as a fuel. The results obtained by this method of firing are gratifying and show a decided saving in cost of smelting as compared to grate firing with lump coal.

The furnace, as remodeled, is 124 ft. long by 21 ft. wide, and varies in height from 8 ft. 6 in. at the back end to 5 ft. 7 in. at the skimming end. The general construction of the furnace is similar to that of the other furnaces at this plant. There are no side doors to this furnace, as it was thought that with the present arrangement for feeding no "fettling" or "claying" would be required. The interior of the furnace can be inspected through the burner port holes, after shutting off the burners and giving a few seconds' time for the gases inside the furnace to clear away. The charging is done on either side of the furnace from longitudinal hoppers, extending a distance of 74 ft. from the back end of the furnace. Leading from the hoppers into the furnace are 6-in. pipes, spaced $19\frac{1}{2}$ in. apart, through which the charge is intermittently dropped. The charge is kept well above the slag line at all times; in this way the side walls are protected and no fettling is needed on this portion of the furnace. The remainder of the furnace requires fettling. After operating for three months, we found that the bricks were eaten into along each side wall from the skimming door back to the point where the charge had been dropped. The depth of this cutting away was 8 in. close to the front end and gradually tapered to zero at a distance of 50 ft., and was greatest on the side of the furnace having the larger flue connection. Hoppers will be put in for the entire length of furnace, from which fettling material will be dropped, to prevent this cutting.

After the run of three months, the roof was in excellent condition. At the back of the furnace the bricks were not cut into at all; at 30 ft.

from the back end they were eaten away 2 in., but at 60 ft. distant they were as when put in. The roof is 20 in. thick. After operating for a while trouble was encountered in tapping the matte. The tap hole was on the east side of the furnace 83½ ft. from the front end. Charging could not be done over the tap hole, or for a distance of several feet on either side; also, owing to the method of charging, matte accumulated in the front of the furnace and could not be completely drained through the side tap hole.

When the furnace was down for fettling in front, it was seen that the calcines fed into the furnace sloped very gently from either side to the center. This, of course, took up the space which in other furnaces is filled with matte and forced the matte to the front of the furnace, and also prevented its being drawn out at the side tap hole. The furnace will not hold more than 50 tons of matte; the other furnaces hold 175 tons. Finally it was decided to tap the furnace at the front. A suitable runway was put in and a tap hole made to the side of and below the skimming door, and all of the matte was tapped therefrom. About 35 tons of matte is tapped per shift. The furnace is skimmed three times per shift. The gases are taken from the furnace through brick flues to either of two batteries of Stirling boilers, each battery developing 650 h.p. One of these flue connections was left as before with a cross-sectional area of 13½ sq. ft., the other flue connection has a cross-sectional area of 40 sq. ft. The smaller flue connection is only used whenever it is necessary to clean the boilers connected with the larger flue. This occurs once a month and lasts for a period of three days, during which time the tonnage smelted is considerably less than when using the larger flue. The following figures verify this statement.

Cross-Section of Flue	Average of	Tons	Ratio
13½ sq. ft.	3 days	405	6.8
40 sq. ft.	3 days before	497	6.7
	3 days after	539	7.3

Conditions which are imperative in order to obtain successful results in coal-dust firing are:

1. The coal, before pulverizing, must be well dried, down to 1 per cent. or less of moisture. This makes it pulverize better and burn more freely. Nothing is lost in drying it separately, as all the moisture must be evaporated before coal (or any other fuel) will burn, and higher efficiency is obtained when the moisture is driven off before using. The furnace itself is the most expensive place in which to dry the coal, as the effectiveness of the whole fire is lowered by the presence of moisture.

2. The coal must be finely pulverized. The increased surface has a direct bearing upon the efficiency obtained in coal-dust firing. It is well to recognize what this increase is. C. D. Demond, Chief Testing Engi-

neer at the Washoe Reduction Works, says: "The approximate diameter of a 1-lb. lump of bituminous coal is 3 in., and its total surface one-quarter of a square foot, while 1 lb. of coal ground so that 95 per cent. will pass through a 100-mesh sieve and 82 per cent. through a 200-mesh sieve has a total surface of 8,000 sq. ft., more or less, depending upon the physical characteristics of the coal, or 32,000 times the area of the single lump."

These figures show that with finer grinding an enormous increase in surface is obtained, and, consequently, an increase in thermal efficiency. At Anaconda the grinding is done so that from 93 to 97 per cent. will pass through 100 mesh and 79 to 82 per cent. through 200 mesh. Coals with higher specific gravity will grind finer in a Raymond pulverizer. N. L. Warford, in charge of our coal-dust equipment, says: "In cement work no better work is obtained when coal is pulverized finer than 95 per cent. through 100 mesh." (This gives from 75 to 85 per cent. through 200 mesh, the percentage depending upon the physical character of the coal.) Coal thus pulverized will contain a high percentage of fine dust practically unmeasurable. As we can burn all the coal thus prepared, there seems to be no good reason for pushing pulverization beyond this point. Coal can be cheaply brought to this condition and the mills able to do this work have large capacity. Higher percentages may be obtained by the sacrifice of capacity, and consequently economy. This standard of approximately 85 per cent. through 200 mesh and 95 per cent. through 100 mesh is a practicable commercial standard and should be maintained.

3. The delivery of the coal and air to the furnace must be controlled so that the proper quantities of each may be secured. Undoubtedly, the proper method of firing pulverized coal is to admit with the fuel the exact quantity of air required for the results to be attained, and to maintain this relationship as long as results wished for are obtained. We know that in order to get the best heat efficiency from fuel a certain amount of air must be provided for combustion. Any air in excess of these requirements dilutes the gases and lowers the temperature; and insufficient air will burn part of the fuel to CO_2 and part of it to CO , which means incomplete combustion.

4. The coal must contain enough volatile combustible matter to give the required combustion. In cement work coal containing as low as 22 per cent. V.C.M. has been used. James Lord, of the American Iron & Steel Co., recommends 30 per cent. as a minimum.

5. The furnace must be properly proportioned, properly equipped, and in good condition.

6. Provision must be made for taking care of the ash formed.

At Anaconda the following equipment is installed. It is larger than is required for one furnace, but was installed with the idea in mind of finally equipping the entire reverberatory plant for coal-dust firing.

The coal from the storage bin is fed into a 30 by 30 in. Jeffrey single-roll coal crusher, where it is reduced to 1 in. maximum size. It is then taken by a belt conveyor to the foot of an elevator passing over a Ding magnetic pulley, which removes any pieces of iron, bolts, etc. It is then elevated and fed by gravity into a 40 ft. by 6 ft. 8 in. Ruggles-Coles drier. The drier consists of two cylinders, the one within the other. Blades of angle iron are fastened to the inner side of the outer cylinder and the outer side of the inner cylinder, so arranged that as the drier revolves the material fed into the space between the cylinders is lifted and dropped on to the inner cylinder and at the same time carried to the discharge end of the drier. The outer cylinder at the discharge end extends beyond the inner cylinder and has a revolving head riveted to it; on the inside of the head are buckets which lift the coal and deliver it out through the central casting. It takes a particle about 30 min. to pass from feed end to discharge end of the drier. At the feed end the inner cylinder is extended beyond the outer cylinder and, passing through a stationary head, is connected with the fire box. The gases are drawn from the fire box by means of a 72-in. Sturtevant fan, forward through the inner cylinder and back through the annular space between the cylinders to the stack. This exhaust fan is placed on top of the fire box and is connected to the drier by means of a 30-in. sheet-iron pipe. The fire-box is fed with lump coal. The capacity of a drier depends upon the moisture in the coal and the speed of the fan. With Diamondville coal, we dry 18 tons per hour. During the month of September, 1914, we used 30 tons of coal to dry 1,984.77 tons of coal.

From the drier the coal is elevated, conveyed by a screw conveyor, and discharged into a steel bin placed above the pulverizer, which is in a separate building from the drier. It is not well to have the pulverizer in the same building with the drier, for the reason that if an accident should occur, causing the coal to overflow, it might be drawn into the fire chamber of the drier and cause a fire, with possible injury to employees.

The Raymond five-roller mill is used. It has an average hourly capacity of $4\frac{1}{2}$ tons. At the top of the main shaft is a rigidly attached spider which rotates with the shaft and to the arm of which the five rollers are pivotally suspended by trunnions carried in bearings in the roller housing. Both upper and lower bearings of the roller journal are provided with long, removable, phosphor-bronze bushings. The rollers are made of cast iron with chilled faces. The grinding is accomplished by the force of centrifugal motion throwing the rollers outward against the steel bull ring. A plow is located ahead of each roller and constantly throws a stream of material between the face of the roller and the ring die.

A fan is connected to this mill from which air is admitted underneath the grinding surface. The material is taken away by the air current as

quickly as it is reduced by the rolls and blown into a Cyclone dust collector placed 20 ft. above the pulverizer. The mill is thus kept free of fine material. The collector is of galvanized steel, cone shaped, and has a return-air pipe leading from it to the housing around the base of the mill. A surplus-air pipe from this return-air pipe relieves the back pressure and is an outlet for any surplus air that may enter with the feed. An auxiliary collector is placed to receive the dust escaping through this surplus-air pipe.

The finished product is discharged through a spout at the bottom of the dust collector, and is taken by a screw conveyor to a bin placed near to and above the furnace.

The coal from the bin is introduced into the furnace by means of an air current delivered through five "burners." The air current is produced by a No. 11 Buffalo fan at a pressure of 10 oz., and, by means of a pipe carrying a nozzle, is introduced into a 6-in. pipe leading into the end of the furnace. The coal dust, fed from the bin by a screw conveyor, drops upon this nozzle, which acts as a spreader, and is mixed with the air and taken into the furnace. A secondary air supply is obtained through the port holes through which the burners are projected into the furnace. These port holes are each 12 in. in diameter, which leaves an annular space 3 in. wide around each of the 6-in. pipes. By means of suitable dampers encircling the burners, this secondary air can be regulated. Another source of secondary air is through four openings between and above the burner ports, the size of the openings being regulated by putting in or taking out brick. The amount of coal fed is determined by the speed of the screw, which is easily regulated by a Reeves variable speed regulator. The entire grinding, conveying, and bin system, from the drier to the burners, is air tight, as far as practicable, with the result that the entire plant is extremely clean and free from dust.

The advantages of this method of firing are:

1. An increase in tonnage, which makes it possible to smelt the necessary tonnages with fewer furnaces.

2. The efficiency of this method is much greater than that of burning coal on grates. In burning coal on grates, there is a loss in transferring to the hearth the heat generated in the fire box. This does not occur in coal-dust firing, where the combustion occurs over the hearth itself. Another point is that none of the heat is lost in gratings from the fire box. In our practice of grate firing, this loss amounts to something like 10 per cent. of the fuel values. The efficiency is greater, due to a uniform temperature being maintained. There are no fluctuations in temperature due to firing cold fuel or to grate cleaning, or to the personal equation of the men firing the furnace.

The superintendent before leaving his work can adjust the feeding of

the coal, and by securing this adjustment with a padlock he can be certain that a definite amount of coal will be burned during his absence. If, for any reason, the furnace men need to shut off the burners, they can do so by releasing a clutch; when this clutch is re-engaged, the burner resumes firing at the original adjustment.

3. Higher temperatures than usual can be produced and maintained because the quantity of coal burned can be easily increased, and also, because there is less excess air to be heated. In grate firing 100 per cent. excess air is used, while in dust firing only 25 per cent. excess air is required. There being a smaller volume of gases, they will remain longer in the furnace, and, consequently, give up more heat to the charge.

4. Less draft is required because there is no bed of coal through which air is drawn. This is an advantage in that less cold air is drawn into the furnace through the side walls.

Tests have been run on Lochray coal, Bear Creek coal, and Diamondville coal. The coals analyzed as follows:

	Moisture	V.C.M.	F.C.	Ash	B.t.u. Dry	B.t.u. Wet
Lochray.....	8.0	29.3	41.8	20.9	10,350	9,730
Bear Creek.....	9.0	35.5	43.4	12.7	11,500	10,580
Diamondville.....	5.6	41.4	44.9	8.1	12,960	12,470

	Length of Test, Days	Tons Smelted per Day	Fuel Ratio	Content of Slag				Matte Cu.	Analysis of Ash			
				SiO ₂	FeO	CaO	Cu.		SiO ₂	FeO	CaO	Cu.
Lochray.....	12	409.4	5.38	37.8	42.1	5.0	0.41	39.5	39.8	13.4	5.6	1.7
Bear Creek...	18	406.7	5.57	38.8	40.8	4.7	0.42	38.9	38.5	11.5	7.2	1.8
Diamondville	30	475.8	7.24	38.1	40.8	5.2	0.42	39.8	36.5	10.3	7.0	3.1

Screen Analysis of Pulverized Coal

	Per Cent. +100	Per Cent. -100	Per Cent. +200	Per Cent. -200
Lochray.....	2.3	97.7	12.1	85.6
Bear Creek.....	4.4	95.6	21.8	73.8
Diamondville.....	6.1	93.9	18.5	74.4

Comparison of Work of No. 7 and No. 8 Reverberatory Furnaces, September, 1914

No. 7 Furnace using Diamondville coal, grate fired.

No. 8 Furnace using Diamondville coal, dust fired.

Furnace	Tons Smelted per Furnace Day	Total Tons Smelted	Tons Coal	Fuel Ratio		
					Excluding Drier Coal	Including Drier Coal
No. 7	250.96	7,260.31	1,870.94	3.88		
No. 8	475.75	14,272.52	1,984.77		7.19	7.08

The ash gave very little trouble. The flue connection between furnace and boiler is cleaned once per day, requiring the labor of two men from 4 to 6 hr. each day. We found the flue was easier to keep clean when using coal containing 22 per cent. ash than when using coal containing 9 per cent. ash; this difference was due entirely to the physical characteristics of the ash. In the first case the ash was light and fluffy, while in the second case it tended somewhat to sintering. Approximately, one-half of the ash of the coal floats on top of, or is absorbed by, the slag, and does not noticeably interfere with the work of the furnace. For the month of September we took from the flue 85.54 tons of ash and flue dust. This material assayed as follows:

Per Cent.	Ounces Ag	Ounces Au	Per Cent.	Per Cent.	Per Cent.
Cu.	Per Ton	Per Ton	SiO ₂	FeO.	CaO.
3.1	5.3	0.019	36.5	10.3	7.0

Very little of the ash goes into the boilers. The boiler tubes are cleaned no oftener than are those connected to grate-fired furnaces.

Samples of gases were taken at about the center of furnace and analyzed as follows:

	First Sample	Second Sample	Third Sample
	Per Cent.	Per Cent.	Per Cent.
CO ₂	16.0	15.0	15.0
O ₂	2.0	3.5	3.5
CO.....	0.0	0.0	0.0

Temperatures are:

40 ft. from back end 2,250° to 2,430° F., average, 2,353°

Flue..... 1,595° to 1,700° F., average, 1,645°

The labor employed in the pulverizing plant is more than normal on account of not using an economical unit. The drier is run 8 hr. per day and requires three men to shovel coal to crusher and one man to tend the

drier. With mechanical feeding of crusher, four times as much coal can be dried at the same labor cost. One man per shift of 8 hr. tends the pulverizer. When more pulverizers are put in one man can tend two of them.

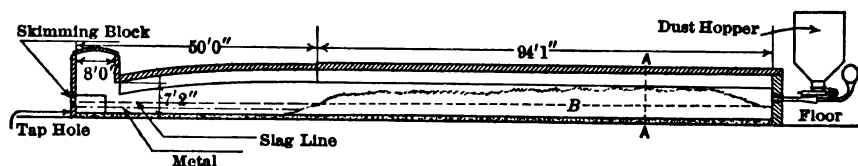


FIG. 1.—LONGITUDINAL SECTION OF COAL-DUST FIRED REVERBERATORY FURNACE.

The repairs as yet are almost *nil*. In time, as the elevators, conveyors, etc., need replacement, repairs will cost something but will never be very high per ton of coal prepared. The electric power for drier, pulverizer, fans, elevators, and conveyors totals 125 h.p. per month.

On account of not working an economical unit a fair average cost of

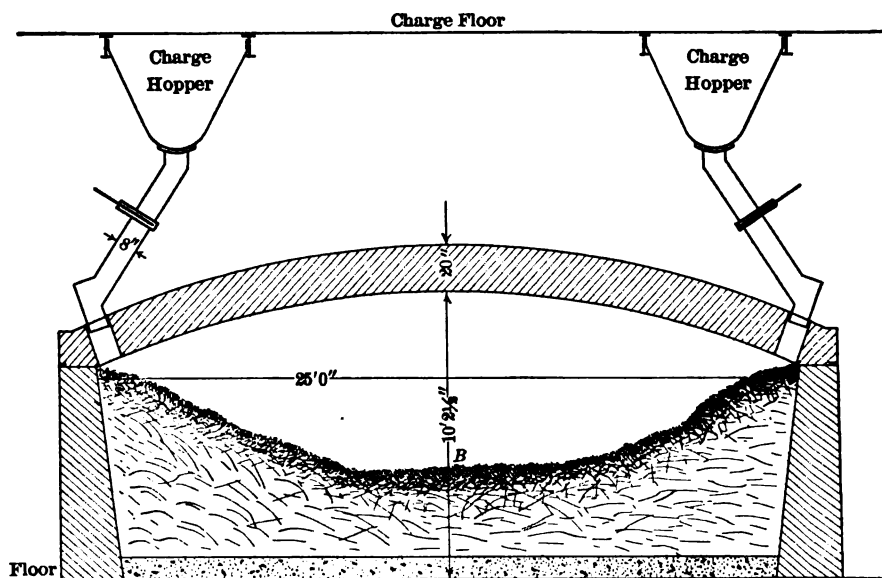


FIG. 2.—CROSS-SECTION OF FURNACE ON LINE A-A OF FIG. 1, SHOWING SYSTEM OF CHARGING.

preparing the coal cannot be given. The general opinion is that it can be done for 35c. to 40c. per ton of coal. This means from 5c. to 7c. per ton of charge smelted. There are three men per shift of 8 hr. doing the skimming, tapping, and charging of furnace.

The management of the works are now planning a furnace of larger

dimensions. Profiting by the work done by No. 8 reverberatory, a few changes in construction will be made. This furnace, shown in Figs. 1 and 2, will be 144 ft. long by 25 ft. wide, inside dimensions; 9 ft. 3½ in. high at the back and 6 ft. 6 in. at the front; flue area, 48 sq. ft. It will have hoppers for charging calcines and fettling material on both sides for its entire length. The tapping of matte will be from the front of furnace. The pipes leading from the charge hoppers will be enlarged to 8 in. The skimming plate will be 12 in. higher than in the other furnaces, thereby giving a larger reservoir for the accumulation of matte. The top of the skimming plate will be 24 in. in height above the tap hole. The general construction will be as before. No changes are to be made in the machinery for preparing and delivering the coal to the furnace.

The coal used at present (Oct. 16, 1914) is not as finely pulverized as heretofore. The following is an average for the past week:

Per Cent. + 100	Per Cent. + 200	Per Cent. - 200
7.5	21.1	71.4

No difference is noticed in the work of the furnaces.

The efficiency of this method of firing is improving so rapidly that a paper is almost out of date before it can be read. New records for tonnage and fuel ratio are made almost every day. The average tonnage per day for the past week was 542.7, with a fuel ratio of 7.50.

I wish to express my thanks to all who assisted me in preparing these notes.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Cloncurry Copper District, Queensland

BY W. H. CORBOULD, CLONCURRY, QUEENSLAND, AUSTRALIA

(New York Meeting, February, 1915)

THE township of Cloncurry is situated in the northwestern part of Queensland, about latitude S. $20^{\circ} 42' 53''$ and longitude E. $140^{\circ} 30' 25''$. Townsville is the port through which all the trade passes and is about 470 miles east-northeast of Cloncurry township.

The Cloncurry copper district covers an area about 200 miles long by 40 miles wide. Unfortunately the mines are very scattered over this area, there being no large groups of mines.

Mount Elliott mine is situated about 70 miles south from the township of Cloncurry, and Mount Oxide mine about 150 miles north of Cloncurry.

The district has been known for the last 35 years, and in the early stages a large amount of capital was expended with little to show for it; but within the last four years the railway has been extended from Townsville to Mount Elliott and about 40 miles north of Cloncurry, which has made possible the treatment of the ores of the mines on this extension.

The Mount Elliott company within the last few years has smelted and converted on its mine 169,688 tons of ore for 20,554 tons of blister copper, containing 20,377 tons of copper, 35,642 oz. of gold and 28,841 oz. of silver, at a cost of about £33 per ton of blister copper, f.o.b. Townsville. This cost included all development, wages, salaries, general management, office, agency, legal expenses, freight, etc. The gold and silver contents would practically pay for freight to Europe, refining, realization charges, etc.

The costs of supplies are as follows: Firewood 15s., coke, 75s., mine timber, 54s., and coal, 60s. per long ton; explosives: dynamite and gelignite, 50s. and blasting gelatine, 65s. 6d. per case.

MOUNT ELLIOTT MINE

The pay shoot of the orebody outcrops at the surface for about 200 ft., and lies between two slate walls about 40 ft. apart, having a general

underlie down to the 180-ft. level of 20° from the vertical; below that depth the ore channel widens out to represent an inverted funnel. The general strike is northwest at the surface, and corkscrews with depth until at the 500-ft. level the strike is nearly due west. The dip follows the strike, and from the surface to the present deepest level is at an angle of approximately 15° to 20° . The copper occurs in oxidized form from the capping down to the 230-ft. level, where the zone of secondary enrichment is encountered, in which the copper is in the form of friable, altered or sub-sulphides. This zone is from 50 to 180 ft. wide, overlying the primary ores, which consist chiefly of chalcopyrites in hornstone. These continue in depth to the present deepest level, 600 ft. from surface.

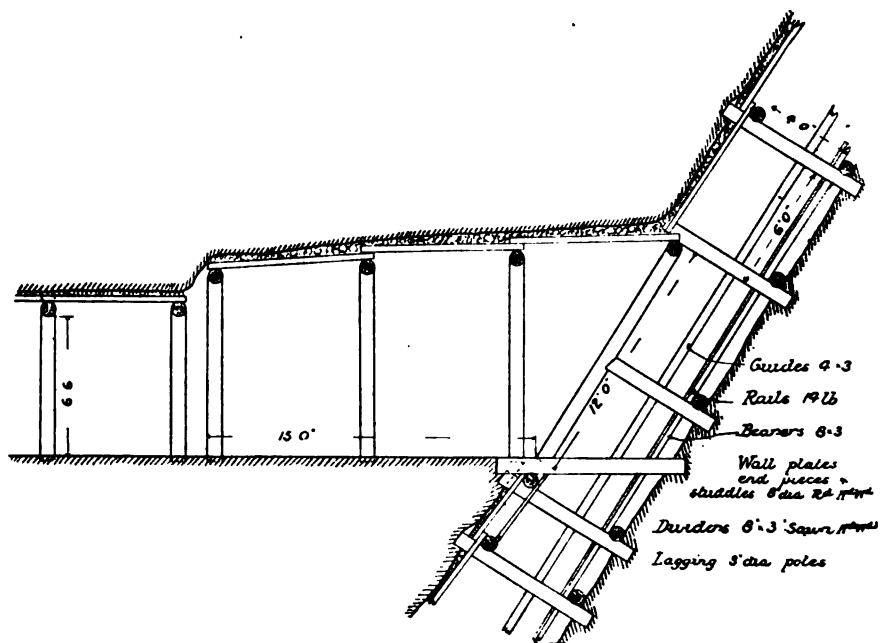


FIG. 1.—MAIN SHAFT AT MOUNT ELLIOTT MINE.

The main hauling shaft (Fig. 1) is sunk in the foot-wall country at an inclination of 30° from the vertical, and is divided into three compartments, two hauling and one ladder way.

The shaft is secured with frame sets cut from 8-in. round timber, spaced 6-ft. centers, and lagged with small poles about 3 in. in diameter.

Each hauling compartment is laid with a track of 14-lb. rails, 18-in. gauge, on 8 by 3 in. continuous bearers of sawn hardwood. The rails are fish plated and spiked to the bearers with 3-in. "dog spikes."

Levels are opened at 120, 180, 280, 400, and 500 ft. vertical depth, and stations cut, from which the main crosscuts go out to the orebody.

(The outcrop was mined by opencutting to a depth of 60 ft.) Drives were put through the orebody in the Nos. 1 and 2 levels for the full length of the pay shoot, with crosscuts to both walls at regular intervals, the main levels being subsequently driven in the foot-wall country.

At the No. 3 level the channel, which has an area of 180 by 150 ft., was contoured, the drives being kept just on the fringe of the lode material, with intervening drives and crosscuts dividing the area into convenient blocks for stoping.

All drives are timbered with sets and covered with split slabs 6 in. by 3 in. by 7 ft. long. In places where the weight is excessive the sets are cribbed. All timber for sets is round ironbark, one size being used

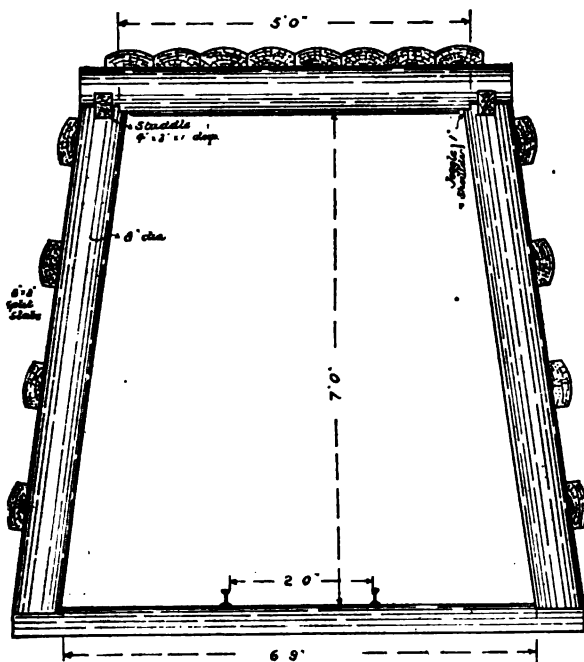


FIG. 2.—DRIVE SETS AT MOUNT ELLIOTT MINE.

throughout the mine. Fig. 2 shows the sets employed, with the covering slabs in position.

No. 4 level (driven in primary ore) has one main drive through the center of the lode, with contour drive on hanging wall and crosscuts to the foot wall. This level has an area of 300 by 170 ft., but only a very small portion has so far been operated by stoping.

Connections between levels and surface are made by raising and winzing, with or without timber, as occasion demands; the size of winzes is 7 by 4 ft., and when timber is used hardwood logs joggled at the ends are employed, sets being placed close. Fig. 3 shows a timbered winze.

All stoping is back or overhand. Above No. 3 level during the progress of stoping numerous cavities (locally termed "vugs") of various dimensions were encountered. The shock from blasts in their vicinity caused these to cave before they were located, with the result that a considerable portion of the stoping carried on above the No. 3 level is through crushed ground, which is particularly heavy, carrying the whole weight in some sections right to the surface.

The square-set system is employed from No. 3 level up, with close filling. The sets are of the same dimensions as those shown in Fig. 4, and are cut from round ironbark logs framed at the saw bench.

The ground is attacked in small sections of about 30 sets area, which are carried to the level above, when the intervening pillars are recovered.

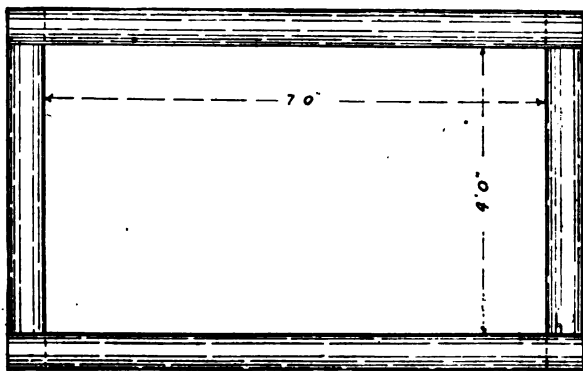
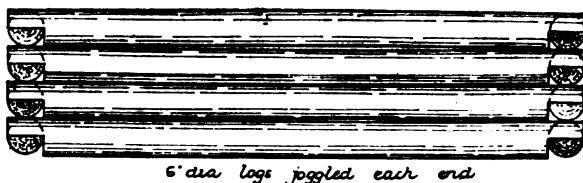


FIG. 3.—TIMBERED WINZE AT MOUNT ELLIOTT MINE.

Where the lode is much broken and consists of loose boulders the "boom" is used to support the points of the back laths while putting in the square sets.

In the friable sub-sulphide zone little drilling or blasting was required, the ore being easily broken with the pick; and in many instances the chief difficulty of the miner was to prevent "runs" of ore, which would leave the hard overlying oxidized ore swinging, resulting in subsequent caving and intermixing with wall rock.

The workings below No. 3 level being in the hard primary ores, the backs are practically self-supporting, so that open stopes are employed, with "flat backs;" the only timber used in these stopes is an occasional

bulkhead to support any loose pieces that may come off from heads and floors that traverse the orebody. Here stopes are carried about 8 ft. high in the face, and since so far only the high-grade sulphides have been treated, the lower-grade material is left in the stope for filling. This filling will come in for treatment in a scheme at present under consideration.

Ore passes are "logged" up through the filling about 20 ft. apart as

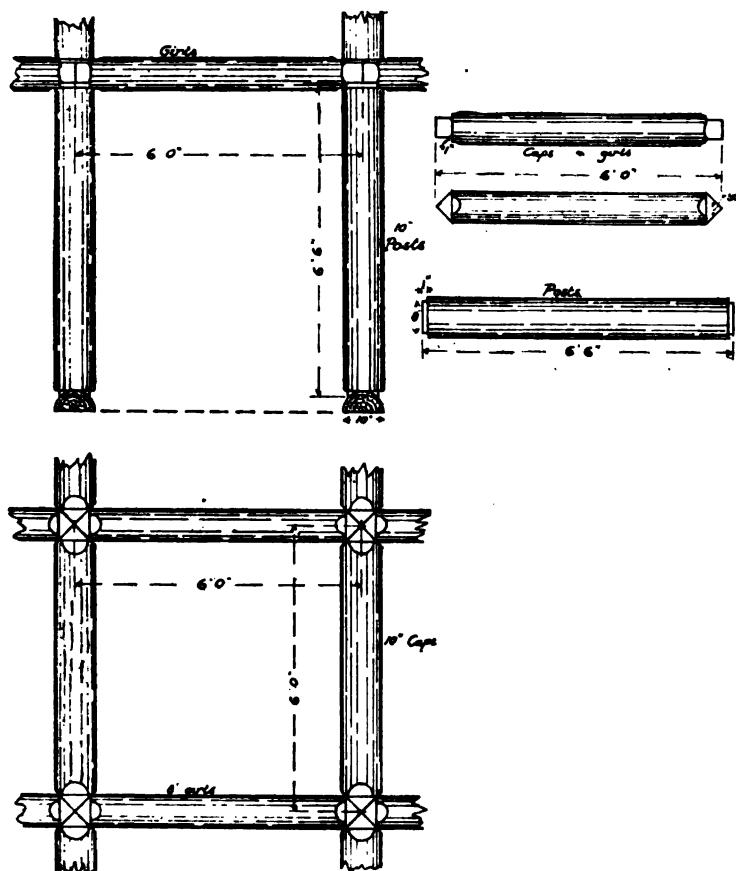


FIG. 4.—SQUARE SETS CUT FROM ROUND IRONBARK LOGS.

stopping proceeds. They are 4 ft. square, inside measurement, and timbered with 6-in. hardwood logs in the same manner as a timbered winze.

The ore is trammed, on the levels from the chutes to the shaft, in trucks of 16 cu. ft. capacity, which contain when filled 16 cwt. of sulphide ore, or from 14 to 15 cwt. of oxidized ore. Tracks are of 14-lb. rails,

laid on wood sleepers placed 3 ft. apart, the grade being kept at 1 in. in 10 ft.

In all development work and sulphide stopes piston drills, $3\frac{1}{4}$ in. in diameter, are used. These are set up on vertical bars and drill holes up to 9 ft. in depth. In stoping above No. 3 level the drilling is done by Ingersoll stoping drills, using both hollow and solid steel, and drilling holes up to 3 ft. 6 in.

In all work underground the nitroglycerine compounds, gelignite and gelatine, are used in plugs 1 in. in diameter and 6 in. long, the former being used for stoping work and the latter for development.

The mine is kept unwatered by a motor-driven, three-throw plunger pump, lifting the water from No. 5 level to the surface, against a head of 550 ft.; air-driven duplex pumps are installed as reserve. The pumping of only 15,000 gal. per 24 hr. keeps the mine unwatered.

Mechanical signal lines, operated by levers at each station, are used for the engine driver, with electric return signals for the platmen.

SELWYN MINE

The Selwyn mine is situated about 20 miles northwest from Mount Elliott. The orebody is narrow, outcrops at the surface at one end of the property, and lies between two defined slate walls. The inclination is about 40° from the vertical in an easterly direction.

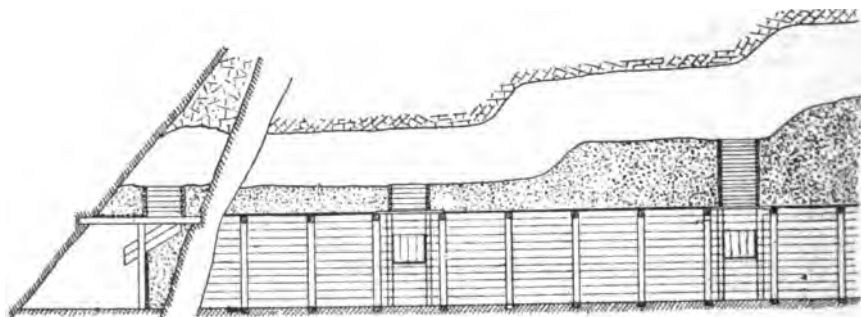


FIG. 5.—MAIN DRIVE AT SELWYN MINE.

The pay shoot, so far opened up, has a northerly pitch and strike. The average width of the lode is about 24 in., widening out in places to 5 ft. The copper occurs as hard oxides and carbonate, being confined strictly to the lode between the walls, no impregnation of the wall rock being noticeable.

The shaft is sunk on the orebody, following the foot wall; size of shaft, 8 by 4 ft. inside timbers. Frame sets of 8-in. round timber are used. The shaft is not divided, a separate shaft being provided for a traveling way.

No. 1 level was opened up 120 ft. from the surface, where the ore channel was 5 ft. wide. The drive is timbered with the "balanced cap," as shown in Fig. 5.

Stoping is overhand, and "resuing" is employed. By stripping the hanging wall of the soft slate, the ore is easily broken, and the stopes filled with the mullock gained during the operation. By this method the ore is kept remarkably clean, and a minimum amount of waste is sent away with the ore. No timber is used in the stopes as supports, but passes are "logged" up through the filling, as stoping proceeds, local timber being used for their construction.

This property is not connected with the smelter by rail. The ore has to be hauled by road traction, 14 tons being transported per trip. At the time of writing the mine is being explored at depth by diamond drilling.

CONSOLS MINE

The Consols mine was purchased by the Mount Elliott company to provide sulphides for Elliott ore. It is situated at Frieze land, about 20 miles northwest of Selwyn.

The orebody outcrops at the surface for nearly the full length of the claim, and carries copper at the surface in varying quantities all the way, but so far not of profitable grade.

The lode varies in width, and has been proved upward of 60 ft. in places. It occurs between two slate walls having an underlie of 10° from the vertical. It is composed of "white iron," quartz, schist, and black oxide of copper, the value being in concentrated form along the foot wall for a width of from 18 in. to 15 ft., where the ore mined has assayed between 7 and 9 per cent. copper. The remaining portion of the lode is low grade generally, with splashes and veins of high-grade material occurring throughout its entire width.

The shaft is sunk vertically in the foot wall, about 10 ft. back from the outcrop; its dimensions are 12 by 4 ft. inside timbers, divided into three compartments, two hauling and one ladder way.

"Box set" timbering is employed, cut from 8 by 2½ in. hardwood, having the wall plates joggled to take end and center pieces. The center pieces are put in close, while the wall plates and end pieces have a 2-in. rising chock between each set down to the 250-ft. level, below which point the shaft timbering is close with a 2-in. chock every five sets.

Guides of 4 by 3 in. hardwood dressed timber are employed. These are secured to the center pieces with ¾-in. diameter wood screws. The guides are put up with butt joints secured by ½-in. diameter iron dowel pins inserted 3 in. into each end. Fig. 6 illustrates the shaft at the No. 3 plat. Stations are cut at 150, 250, and 350 ft. from the collar. Station sets are cut from 9 by 9 in. hardwood timber as shown.

The drives are run along the foot wall in the orebody. The drives are 7 by 7 ft. inside timbers, and are secured with "tunnel" sets of 8-in. round timber. The sets are placed 6-ft. centers, and covered with split slabs 8 in. by 3 in. by 7 ft. long. Fig. 7 illustrates the drive in use.

Winzes are put through from level to level, and connected with the surface for filling and ventilation. The winzes are 7 by 4 ft.; secured with 6-in. logs where necessary.

The stopes, as previously stated, vary in width from 18 in. to 15 ft.;

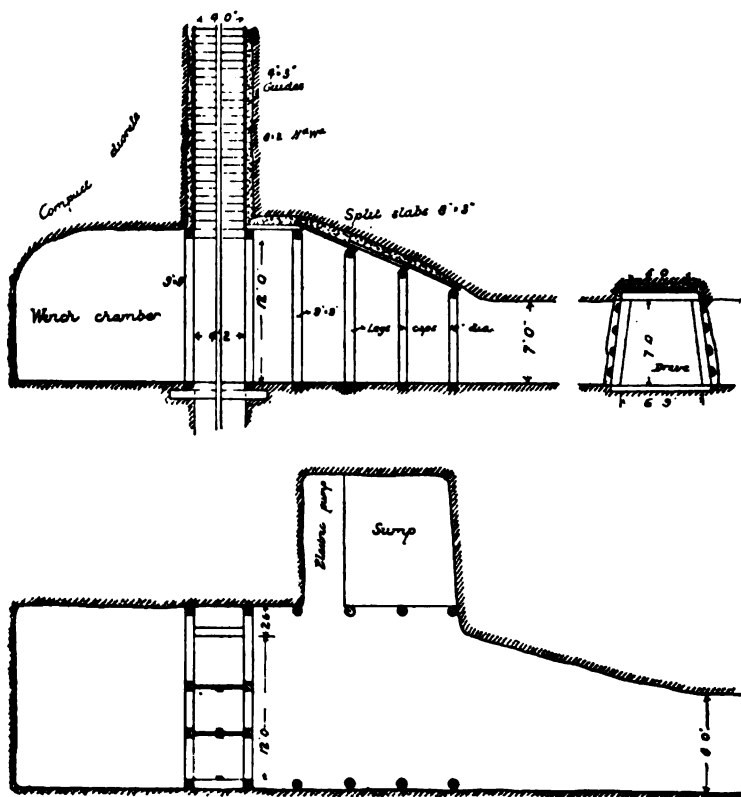


FIG. 6.—SHAFT AT CONSOLS MINE.

by means of which all the 7 to 9 per cent. ore along the foot wall is taken out. A modification of the "rill" system is employed. This method not only reduces the amount of handling when the ore is broken, but facilitates ventilation, which is highly important in this case on account of the amount of sulphur liberated when the ore is broken. When the stopes exceed 4 ft. in width the backs are secured with stulls or light sets; narrower places are allowed to swing, the wall being secured temporarily with stull and head board until the filling is put in.

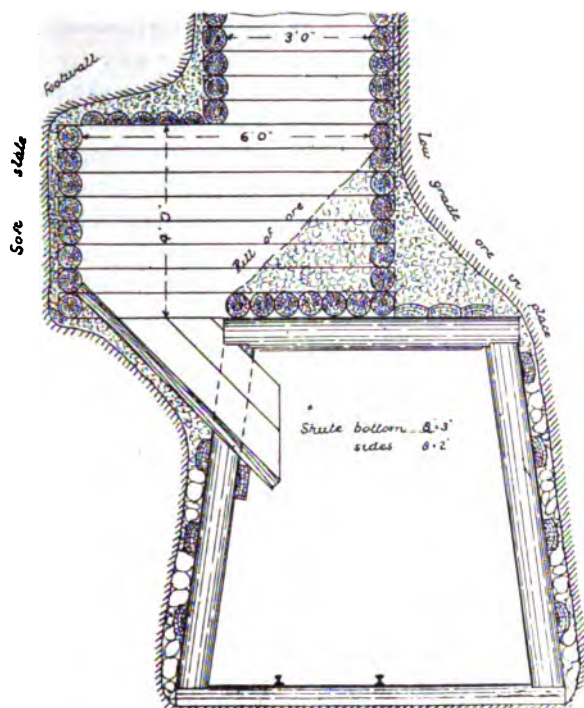


FIG. 7.—ORE PASS AT CONSOLS MINE.

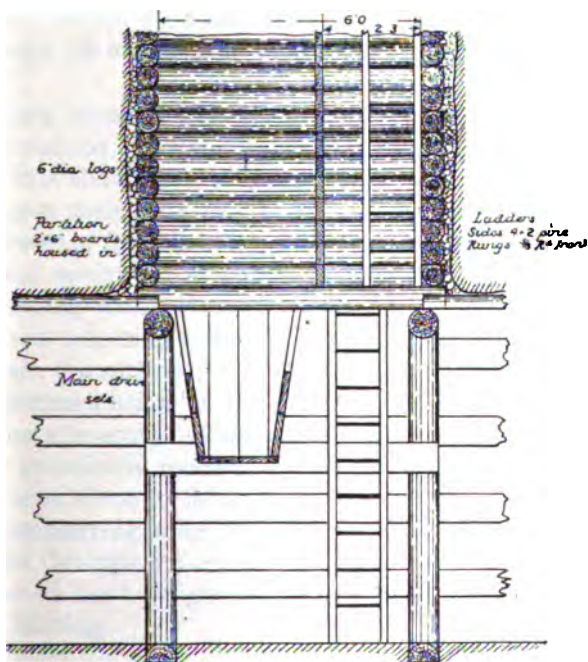


FIG. 8.—ORE PASS AND LADDERWAY AT CONSOLS MINE.

Passes are brought up every 25 ft. as stoping proceeds. The passes are 6 by 3 ft., divided into an ore and ladder way by a partition of 2-in. boards let into the side logs. Fig. 8 illustrates the passes in use.

The stopes are filled with mullock from the surface and waste produced from development work. Very little handling is done in the stopes; the angle of repose of the filling as run in determines the angle of the backs for stoping, except when approaching the level above.

Machine drills are employed in most development work, while in the stopes down to the 250-ft. level the ore is easily bored with hand augers, and considerable ground is broken with picks.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Origin of the Louisiana and East Texas Salines

BY EDWARD G. NORTON, CHICAGO, ILL.

(New York Meeting, February, 1915)

THE salt deposits of the Mississippi Embayment region present a problem of origin so genetically related to the larger problem of the stratigraphy and structure of the region that a discussion of the one necessarily involves or assumes an understanding of the other. For this reason, I shall summarize briefly our knowledge of this region, emphasizing the facts related to the subject of this paper. In so doing I shall make liberal use of the reports of the National and State surveys.

Geographically that section of the United States known as the Mississippi Embayment region represents a great V-shaped area, extending from near Cairo, Ill., to the Gulf of Mexico, and including Louisiana, eastern Texas, all that part of Arkansas bordering the Ouachita and Ozark mountains, western Tennessee, Mississippi, and a part of Alabama.

This great section Professor Harris has described as structurally a pitching trough, its medial line being well represented by the Mississippi River.

Outcropping along the border of this area are the Cretaceous and Tertiary formations, unconformably laid down on the hard Paleozoic rock sheets that show more or less plication, deformation, and faulting, depending upon their proximity to the nearby major uplifts.

East and west of the Mississippi River the Cretaceous and Tertiary beds dip river- and coastward at angles greater than the slope of the land. "Waters entering along the outcropping edges of the Tertiaries and Cretaceous and following bedding planes southward are soon thousands of feet beneath the surface."

As the sedimentation of the region progressed, faulting and slips occurred along planes of pre-Cretaceous weakness. Whether these movements were genetically related to the loading of this V-shaped area, as Professor Harris seems to think, is a question still to be settled.

The accompanying map, Fig. 1, was taken from *Bulletin No. 7* of the Louisiana Geological Survey. It shows the regional distribution of the salines, as well as an alignment of the domes, which has been assumed to indicate faulting.

Geologic History of Northern Louisiana and Southern Arkansas

Geologic subdivisions		Characteristic activities	
		Deposition	
		Thickness	Character
Quaternary.	Recent.	Feet. 0- 20 +	Veneer of sand, silt and clay on flood plains.
		0- 25	Abnormal deposits of silt in Red River Valley resulting from the obstruction by the "great raft."
	Pleistocene.		Formation of natural mounds.
		Port Hudson formation. 0-200	Marine deposits on the coast and fluviatile deposits in the river valleys, partly filling the broad valleys developed in the preceding erosion cycle.
Tertiary.	Pliocene.		Rearrangement of surficial sands and gravels at new levels as erosion progressed.
		Lafayette formation. 40- 50	A mantle of silt, sand, and gravel spread by combined marine and river action over the relatively even surface of the Coastal Plain and in the tributary valleys.
	Miocene.		
	Oligocene.	Fleming clay. ± 260	Green calcareous clays, with a few brackish-water fossils.
		Catahoula formation. 1,000-1,200	Near-shore deposits; sandstones occasionally quartzitic, and green clays, with fresh-water shells and land plants.
		Vicksburg formation. 100- 200	Limestones and calcareous, somewhat lignitiferous, clays, containing marine shells.
	Eocene. ^b	Jackson formation. 200- 550	Highly fossiliferous shallow-water marine sandy calcareous clay.
		Cockfield member of Claiborne. ^c 400- 500	Lignitiferous sands and clays, with land plants.
		Claiborne formation. 200- 500	Fossiliferous sandy clay, containing shallow-water marine shells.
		Sabine formation. 300- 900	Lignitiferous sands and clays, with plants and occasional beds of marine shells.
		Midway formation. 20- 260	Limestones and black calcareous clays.
Cretaceous.	Gulf series (upper Cretaceous).	Arkadelphia clay. 300- 600	Black laminated clays, with marine fossils in lower portions.
		Nacatoch sand. 100- 160	Sand, with occasional calcareous quartzitic layers containing marine shells.
		Marlbrook formation. 150- 750	Very calcareous clay, with marine fossils.
		Annona chalk. + 100	White chalk.
		Brownstown formation. 150- 500	Blue calcareous clay, with marine fossils.
		Bingen formation. Sub-Clarksville sand. ^d 50- 100 Eagle Ford clay. ^d 0- 600 Woodbine formation. + 500	Water-bearing sand. Blue calcareous clay. Lignitiferous sands and clays, with plant remains (Bingen formation).
	Comanche series (lower Cretaceous).	Washita group. 0- 400	Dark calcareous clays, with clayey limestone beds.
		Goodland limestone. 25- 30	Massive white limestone.
		Trinity formation. 500- 600	Yellow, somewhat argillaceous peck sands, changing to the east to calcareous clays, with thin beds of limestone and gypsum.

Pre-Cretaceous. An immensely long period composed of the following principal stages in the order leveled these folds. On this planed-off surface the Cretaceous beds were deposited.

^a Normal thickness in northern Louisiana and southern Arkansas not known because of the widespread and irregular deposition. In southern Louisiana these beds are much thicker than here given.

^b The Jackson, Claiborne, and Sabine formations, which are fossiliferous and distinct in central Louisiana, grade into lignitiferous beds containing no distinct fossils as they go northward. In the region under discussion the fossiliferous Jackson limits this lignitiferous complex above. Still farther north, however, the Jackson also grows lignitiferous and merges with the rest. The Midway, likewise, in the

Degradation	Deformation
General degradation of the hill lands. Along Red River in Louisiana the resurrection of buried channels and the drainage of lakes produced by the "great raft." On the Sabine the partial wearing out of shoals produced by the recent movement of the Angelina-Caldwell flexure. Partial removal of valley fillings and production of present flood plains and principal terraces.	A slight upward movement at the west end of the Rockland-Vicksburg flexure is producing rapids on Sabine and Angelina rivers. A recent movement of 25 feet along the line of the Red River-Alabama Landing fault has resulted in the swamping of Ouachita River Valley to a point above the mouth of Bayou Moroin Arkansas
Long and complex period of erosion, with the land 100 feet higher than to-day, in which the formations of the Coastal Plain were profoundly dissected and the major features of the present topography produced. A period of erosion, probably composed of several stages, in which the Coastal Plain in this region was essentially base-leveled.	After the main development of the Angelina-Caldwell flexure the beds were faulted along a line extending from a point near Denison, Tex., through Alabama Landing, Union Parish, La. The downthrow of this fault is to the north and the break approximately 600 feet The low fold which extends from the vicinity of Angelina County, Tex., to a point north of Vicksburg, Miss., and which is now a line of weakness, began to develop in late Oligocene or early Miocene time. North of this line the older beds are now nearly horizontal; to the south they dip at a rate of from 35 to 150 feet per mile.
Beds separated by a pronounced break in the fauna, which is, at present the only indication of a very serious break in sedimentation. The Bingen and Woodbine sands indicate deposits very near shore and the fauna changes sharply at this point, but there is no evidence of a pronounced unconformity or a land period of great length.	The domes developed during late Cretaceous and early Eocene time show a slight movement in post-Claiborne time, but the amount is very small when compared with the initial movements. The great north-and-south fault of the Coastal Plain of Texas (the Balcones fault) developed late in the Cretaceous. In Louisiana peculiar domes or four-sided folds were produced and reached their major development in the late Cretaceous or early Eocene. About the same time masses of igneous rocks of limited area were intruded into the Paleozoic rocks and Coastal Plain beds in southern Arkansas. In central Texas similar occurrences took place as early as the Austin epoch. The Louisiana and northeastern Texas domes are thought to be due to the upthrust of similar igneous intrusions.

given: (1) Deposition, (2) profound folding and faulting, (3) long erosion, which essentially base-

upper embayment region shows a decidedly lignitiferous tendency and may in places merge with the lignitiferous time equivalents of the other Eocene beds.

A group without distinctive marine fossils, probably almost wholly of Claiborne age.

These beds do not outcrop in the Arkansas area, except as they may be represented as littoral deposits in the upper part of the Bingen sand, and therefore need not be considered in wells drilled near the northern edge of the Brownstown formation; but they will be found farther south and must there be allowed for in well calculations.

That some of these fault zones exist, as shown on the map, is quite certain. Especially is this believed to be true of those running northeast and southwest. Others are to a great extent conjectural; and while some doubtless exist others have been assumed without much reason, and seem to impose upon nature a criss-cross arrangement of fault lines of such great regularity as to exceed her tolerance for geometrical design.

The arrangement of these domes in the vicinity of the "Sabine Peninsula" is noteworthy, as is the alignment of the Five Islands.

The stratigraphy and history of this region have been so well summarized by Veatch in *Professional Paper No. 46* of the U. S. Geological Survey that I introduce his table here.

It will be noticed that in this statement the land period represented by the ligniferous sands and clays of the Bingen formation is separated from the ligniferous sands and the shales of the Tertiary by the marine beds of the upper Cretaceous. Penetrating the Cretaceous and Tertiary beds are cores of rock salt several hundred feet in diameter and from 2,000 to 3,000 ft. in thickness. These cores of salt are often capped by a highly crystalline limestone or marble, that is quite as remarkable and difficult of explanation in this alluvial country as is the occurrence of the salt itself. Around them are found deposits of gypsum. The further occurrence of sulphur, dolomite, sulphur dioxide, hydrogen sulphide, hot saline waters, large accumulations of oil and gas, are all peculiar to these localities.

The disturbance of the clays and shales in the immediate vicinity of these deposits indicates faulting; and the alignment of the salines would suggest that their distribution is along such lines of weakness.

If salt in regular bedding exists in Louisiana and Texas it has never been penetrated by the drill. If we assume that it exists thus at depths greater than have been reached, and has been elevated to the surface by an anticlinal development, the assumption is not supported by evidence that such mountain-building forces have been at work. The sediments throughout Louisiana and Texas show little disturbance.

At present there seems to be great unanimity of opinion among geologists that the salt, crystalline limestone, dolomite, and gypsum are secondary, and not indigenous to the formations in which they occur.

The origin of these salines has been the subject of much interesting speculation. An outline of the different hypotheses that have been advanced in explanation of these unique deposits will be found in *Bulletin No. 7* of the Geological Survey of Louisiana.

In investigating them one is impressed by the fact that chemical reactions which appear to be taking place at the surface result in the formation of the same compounds that we find ranging from the surface down to depths of 1,500 or 2,000 ft. Moreover, proximity to the surface seems to favor such chemical reactions, if it does not actually make them possible.

ened by the fact that, in northern Louisiana, cross-bedded Tertiary sands are by no means uncommon.

Since the cores of salt and gypsum farther south presumably penetrate these formations, the questions arise: Have not the cores been formed contemporaneously with the beds? Have not these salines for a very long time presented about the same appearance as now? Is the chemical action that is at present taking place very different from what it has been in the past? Are not these salines the product of the same forces as are now active?

Hayes and Kennedy, in describing High Island,¹ mention the fact that

"At various points about the margin of the island are salt springs, which give rise to the trembling marshes. The water contains much calcium carbonate in solution, which is precipitated by evaporation about the spring vents and forms a crust over the surface of low mounds, being held together by the coarse marsh grasses."

A. C. Veatch² calls attention at Drakes saline to an "outcrop of grey, granular, sandy limestone containing very imperfect plant impressions."

Vaughan has made the following statement: "Near Atlanta in Winn parish, there outcrops a hard, blue limestone, which is traversed by minute fissures. In these fissures a small amount of gold is found." Veatch adds that "this must have been a near shore deposit, for it contains³ the impressions of dicotyledonous leaves."

Limestone containing leaves I would not regard as of marine origin, or as having been formed in the ordinary way. Inasmuch as its occurrence is confined to these salines, the action that is at present taking place at High Island would seem best to explain its origin.

A well section at Rayburn's salt works⁴ is interesting:

- A. Whitish mud of the lick, with ferruginous spots and at its base frequently bearing balls of calcite..... 6 ft.
- B. Siliceous gravel often cemented into a conglomerate by crystallized calcite 6 to 7 ft.
- C. Greyish or white crystalline limestone, horizontally banded, fragile, often covered with 5 to 6 in. of crystallized aggregates of calcite, on a dark banded base of the same..... 6 ft.
- D. Dense, banded gypsum, pure..... 2 ft.

The siliceous gravel that has been cemented into a conglomerate by crystallized calcite is supposed to be of Pliocene age.

It would appear from the literature, as well as from my own investigation, that calcium carbonate, limestone containing plant impressions, gypsum, sulphur, and probably salt are being formed at or near the surface at the present time.

¹ *Bulletin No. 212, U. S. Geological Survey*, p. 122 (1903).

² *Report of the Geological Survey of Louisiana for 1902*, p. 60..

³ *Report of the Geological Survey of Louisiana for 1899*, p. 60.

⁴ *Report of the Geological Survey of Louisiana for 1902*, p. 73.

Inasmuch as our ingenuity is about exhausted in an attempt to explain the occurrence and origin of this same group of minerals underground, the assumption would in a measure seem to be justified that they have for the most part been formed at or near the surface and that these cores of salt and gypsum have been built up contemporaneously with the sedimentation of the region.

The hypothesis that I have formulated in an attempt to harmonize the field evidence that these salines present, either "singly or in combination," is as follows:

The Tertiary salt deposits of the Gulf coast and the Cretaceous salines of eastern Texas and northern Louisiana were initiated by the intrusion of molten rocks into the underlying Paleozoic sediments along lines of structural weakness. These great faults were the sites of frequent downward displacements during the subsidence and deposition of the sands, clays, and littoral marine sediments of the Mississippi Embayment region.

Hot, ascending solutions, containing calcium and magnesium carbonates, sodium chloride, carbon dioxide, with varying amounts of hydrogen sulphide, mingled with the artesian saline waters of the Cretaceous beds. These waters were forced upward to the surface by the hydrostatic head of the region, through channels that were opened by the faulting and movement over these intrusions of igneous rock.

Great deposits of travertine or calcareous sinter, similar to the deposit at Winnfield, La., were formed around the thermal springs that issued from these openings, the sinter continuing to build as long as the hydrostatic head was sufficient to maintain the flow.

If we assume, as we must, a constant hydrostatic head for the artesian saline waters of the Cretaceous beds, and if the subsidence of the Mississippi Embayment region is due, as has been thought, to the increasing weight of accumulating sediments, then the flow from these springs that determined the rate of sinter building must have been closely related to the sedimentation of this section. If, on the contrary, as has been urged, the downward movement and displacements determined the amount of sedimentation by the depression of the drainage below its base level, and the subsequent silting up of its channels and concurrent building of great flood plains, such as are being formed by the Mississippi and tributary streams at the present time, it is apparent that the same secular movements which promoted such an increased accumulation of sediment would, by relatively increasing the hydrostatic head, augment the flow of water from these springs, and so add to the rate of sinter accumulation. In other words, the balance that was maintained between the rate of sedimentation and the rate of sinter building must have followed such a causal relation.

As these great deposits of calcareous dolomitic sinters were built up,

more or less gypsum was deposited in the surrounding marshes by the oxidation of the hydrogen sulphide coming in contact with the oxygenated surface waters, the H_2SO_4 so formed converting the calcium carbonate of the spring waters into the relatively insoluble sulphate.

Contemporaneously with the building of these sinters, sands and clays were deposited around their bases. At times, owing to the sudden increased activity of these springs resulting from downward movement and relative increase of hydrostatic head, the sinter accumulation encroached upon the marsh. At other times the accumulation of sediment encroached upon the sinter.

As the sinter continued to build, coincident with the subsidence and sedimentation of the region, the same excess of carbon dioxide in the ascending waters that prevented a deposition of carbonates in the channel below, attacked and re-dissolved the bottom layers. By the periodic rapid deposition of the sinter above and its slow, constant dissolution below by the carbonated saline waters, open spaces were developed that were carried upward, in which the salt was deposited from ascending solutions that were supersaturated with saline contents by the release of pressure, as well as by the evaporative losses these waters must have sustained at the surface, as the rapid sinter accumulation checked the flow from the springs.

In this way these salt deposits several thousand feet in thickness were built up contemporaneously with the sedimentation of the region and formed under their protective covers of sinter, and so we find them to-day, where they have escaped erosion and have been preserved in their entirety.

The accumulation of sinter around hot springs is relatively so rapid that the periods devoted to active sinter building, immediately following downward displacements of a few feet, will occupy but little time—a few years at most, as contrasted with the much greater intervals of the comparative inactivity or complete repose of the springs.

Such is briefly the origin of these deposits. The great deposit of travertine at the Winnfield marble quarry, and the undoubted salt core of great thickness beneath it, is too suggestive of such an origin for its presence in the midst of lower Claiborne sands and clays to be lightly ignored. Neither should the occurrence of the leaf-bearing limestone, mentioned by Veatch, be overlooked.

The following is the log of a well at Drake's Saline:⁵

	Ft. to Ft.	
Soil, muck, pebbles, logs.....	0	303
Limestone (like marble at Winnfield).....	303	910
Salt.....	910	2,320
Gypsum bed at bottom.....	...	2,342

⁵ *Bulletin No. 429, U. S. Geological Survey (1910).*

The log shows travertine over 600 ft. in thickness and resting upon the top of the salt.

In northern Louisiana and eastern Texas, all the Cretaceous salines are located along bayous, branches, or creeks, the artesian flow of waters from these springs having eroded their own channels.

To illustrate the rapidity of sinter accumulation:

"The travertine of Tuscany is deposited at the Baths of San Vignone at the rate of six inches a year, at San Filippo one foot in four months. At the latter locality it has been piled up to a depth of at least 250 feet, forming a hill a mile and a quarter long and a third of a mile broad. An illustrative illustration of the rapidity with which the travertine may be deposited, is furnished by the Eocene sinter of Sezanne, Marne. This deposit contains hollow casts of flowers which fell on the growing sinter, and were crusted over with it before they had time to wither. As the material thickened round them they decayed inside, but the hardened carbonate preserved an accurate mould of their forms. When hot wax is injected into these cavities, and the surrounding lime is dissolved away with acid, perfect casts of the flowers are obtained."⁶

There is nothing antagonistic to this hypothesis in the stratigraphy of this section of the Mississippi Embayment region, as revealed by numerous well records. Undoubted marine beds occur; and it is quite certain that salt cores with their protective caps of sinter were depressed below sea level during the upper Cretaceous and their spring waters were effectually sealed by the deposition of the marine sediments above. This probably happened to the salt deposits of eastern Texas and northern Louisiana, but could in no way have affected the tertiary salines of the Gulf Coast region, which were doubtless of a later development, although, in all essential details, analogous to the Cretaceous salines farther north.

Some of the salines of the Gulf region were submerged in late Tertiary seas. Well records show at depths of about 1,000 ft. the crystalline limestone that overlies the salt at Spindletop. Other domes, confined to a zone running practically due east and west from Beaumont, probably have had a similar history. As the axis of the folding parallels this direction a sudden downward movement of considerable magnitude no doubt depressed these salines below the Gulf level.

A microscopic examination of the rock salt from Avery's Island tends to corroborate and strengthen the hypothesis formulated to harmonize the field evidence gathered from a collective study of these different salt deposits.

Under the microscope, all the salt crystals examined by the writer reveal included gypsum crystals, with their ends broken or rounded. These crystals, when dissolved out of the salt, often show inclusions of foreign matter, from which the salt itself is relatively free.

The occasional presence of small quartz crystals in the Petite Anse

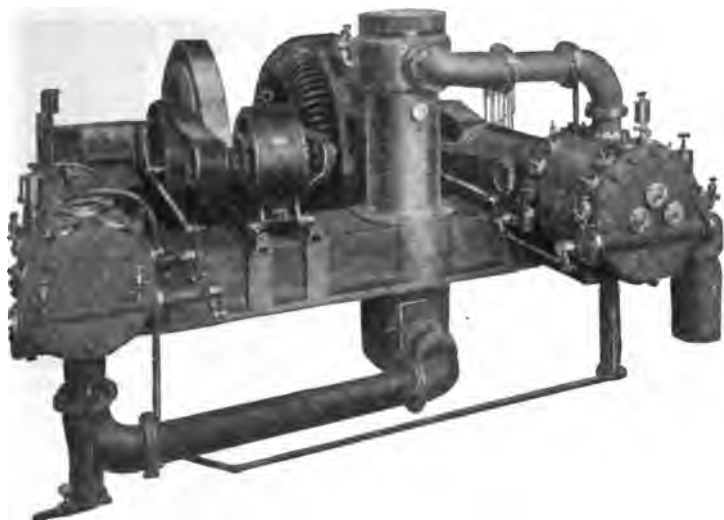
⁶ Geikie: *Text Book of Geology*, 4th ed., p. 476 (1903).

salt containing the same inclusions as were noted in the gypsum, seems to indicate that both were formed under similar conditions.

Since the sinter at Winnfield contains great numbers of these small crystals of quartz, showing, under a magnification of 300 diameters, the same inclusions, or what appear to be so, it is probable that they were derived from the sinter; and their presence in the salt must be ascribed to the downward migration of all insoluble material formed in the sinter by the constant deposition above and solution below. When the salt is broken, a strong odor of hydrogen sulphide is emitted.

The quaquaversal structural features of these domes are no doubt due to faulting and downward displacements. The introduction of hard cores of salt into these planes of faulting contributed greatly to such structural features as characterize these salines.

I have not attempted a detailed criticism of the different theories advanced in explanation of these unique deposits. All are of value, inasmuch as the discussion has eliminated much misconception and has contributed greatly to our knowledge of these salines.



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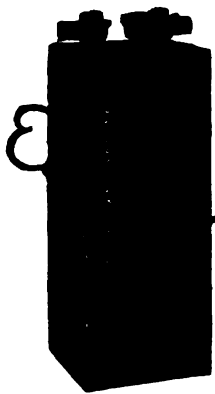
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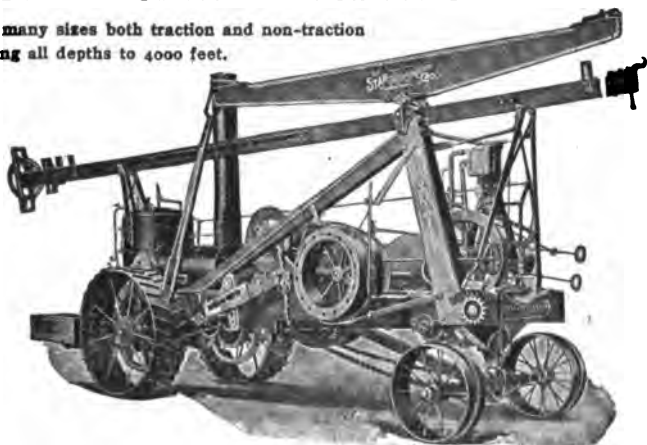
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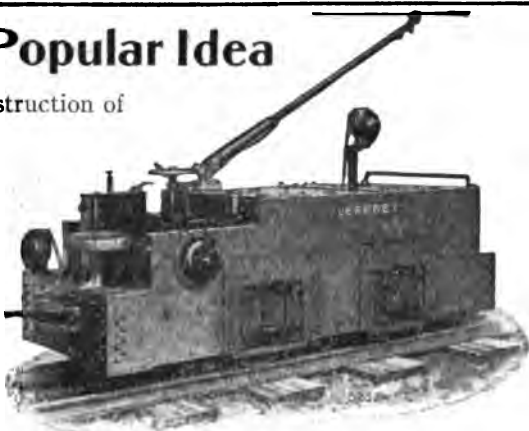
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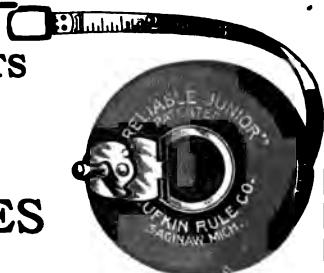
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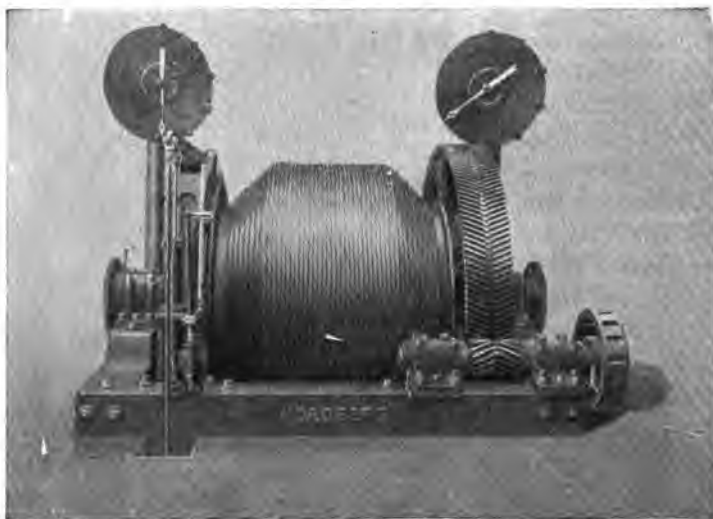
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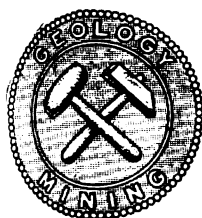
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FEBRUARY, 1915



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29 West 39th Street, New York, N. Y.

PROPOSAL FOR MEMBERSHIP

Mr. _____
(Name in Full)

Occupation_____

Address _____

is hereby proposed by the undersigned, as a _____

of the American Institute of Mining Engineers.

Signatures of three
Members or
Associates.

Place of birth _____ Year of birth _____

*Education, general and technical, when, where and how acquired,
with degrees, if any.*

Dates

Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)

[illegible]

Present position

Signature.

Dated _____ *191* _____

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

Harvard College Library

Aug. 14, 1916.

Bequest of

Erasmus Darwin Leavitt.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 98

FEBRUARY

1915

Published Monthly by the American Institute of Mining Engineers at 212-218 York St. York, Pa., H. A. WISOTSKY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY SROUGHTON, Editor, F. O. PIERCE, Asst. Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$10 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

FACILITIES OFFERED TO MEMBERS AT INSTITUTE HEAD- QUARTERS

Library Service.—Special efforts are made to make the library as serviceable as possible to members who are distant from headquarters and this is accomplished practically by the means of researches, copying and translation facilities, etc. More detail on this matter is given in the section of this *Bulletin* devoted to the library.

Members' Writing Room.—At the headquarters of the Institute, 29 West 39th Street, there is maintained a Members' Writing Room, which is equipped with writing, telephone, telegraph, messenger, dictation and other office facilities. A file of current magazines is also maintained in this room and the Institute's staff will lend its best efforts to serving the members in connection therewith.

Dressing Rooms.—On the ground floor of the Engineering Societies' Building, 29 West 39th Street, are a number of free dressing rooms, where members will find all the necessary facilities. This is particularly intended for members who arrive in New York on early trains or who desire to make a change of clothes for evening entertainments. The building is open from 8 A.M. until 10 P.M. and access may be had at other hours by ringing the door-bell.

BOARD OF DIRECTORS

Meeting of Dec. 18, 1914.—Charles F. Rand, Karl Eilers, and William L. Saunders were appointed a Committee to audit the accounts of the Institute.

The matter of the Committee to investigate the relative merits in metallurgical work of turbo-blowers and the reciprocating blowing engine was laid on the table.

Benjamin B. Thayer was elected representative on the Board of Trustees of the United Engineering Society to succeed James F. Kemp.

Upon the recommendation of the Committee on Membership, 32 Members, 8 Associates, 3 Juniors were elected, and the status of one Associate was changed to that of Member.

Twenty-six members owing dues for from three to six years were dropped from the rolls in accordance with the provisions of the Constitution.

The resignations of 16 members were accepted.

The following resolution on the death of Samuel B. Christy, formerly Chairman of the San Francisco Local Section, was unanimously adopted:

Samuel Benedict Christy, Professor of Mining and Metallurgy in the University of California since 1885, member of this Institute since 1883, its Vice-President in 1891-2, 1907-8, and 1911-12, one of the leaders in the education of mining and metallurgical engineers, prominent in the development of the mineral industry of the Pacific Coast and in self-sacrificing metallurgical investigations for the good of the mining industry generally, was a zealous and strict supporter of all that is good and best in the ethics of the mining and metallurgical professions.

A graduate of the University of California in the class of 1874, a student under William Ashburner and Joseph Le Conte, he gave such early evidence of his professional ability that in 1874 he was made an instructor in his *alma mater*, and in 1885 was made Professor of Mining and Metallurgy. He made the development of the course in Mining at that University his life work, finally building it up until the school, with its own building and large teaching staff, was one of the most important in the University. The education of engineers was his chosen task, and to it he devoted a degree of enthusiasm which is reflected by the two papers on that subject which he contributed with other papers to the *Transactions* of the Institute. He found time for a wider range of activities, however, and his studies of the chemistry of the cyanide process, especially the precipitation of precious metals from cyanide solutions by electrolysis, are classics on the subject. He had recently spent much of his time preparing a monograph covering his studies and work in this field. A native of California, it was natural that his professional interest should turn toward gold and silver, and he contributed many notable papers on these subjects to the *Transactions*, and to other publications.

Largely through his interest in the matter, and as a result of his publications and addresses, courses in mining engineering have been provided in all the State Universities of the West, and the present high standard of education among professional mining engineers has come about. This was fittingly recognized by Columbia University, which conferred upon him the degree of Doctor of Science in 1902.

His long and earnest life was crowded full of achievements for the best interests of the profession to which he was an ornament, and was suddenly terminated by the quiet hand of death. We profoundly regret his loss, but we rejoice in his memory of devoted service, an unperishable inspiration for those who come after. It is therefore ordered that this be spread on the minutes of the Board, and that a copy be sent to his widow. Also that his Biography with portrait be published in the *Transactions*.

It was voted that the date of the Annual Dinner be changed to Tuesday, Feb. 16, and the date of the 110th meeting be changed to Feb. 15, 16, and 17, 1915, instead of Feb. 16, 17, and 18, 1915.

J. F. Pack was appointed a member of the Committee on Mining Geology.

AFFILIATED STUDENT SOCIETY NOTES

The officers of the Mining Society of the University of Pittsburgh are: W. G. Flood, President; C. H. Corbus, Jr., Vice-President; J. H. Clements, Treasurer; and L. O. Shannon, Secretary.

Victor Menaglia has been elected President and Cassius Parmalee Secretary and Treasurer of the Mining Engineering Society of the State College of Washington. The Society holds meetings semi-monthly, on alternate Wednesdays, at which papers presented by the members are read, or addresses are delivered by members of the faculty or by visitors who are interested in the mining industry.

An especially successful meeting of the Kentucky Mining Society, State University of Kentucky, was held on Dec. 1. The program comprised an address by the President of the University, Henry S. Barker, LL. D.; a paper on Mine Management by Hywel Davies; and a paper by Joseph Reed, Assistant State Inspector of Mines, on The Duties and Work of a Mining Engineer. Students from the other Engineering Colleges of the University were present, and refreshments were served at the conclusion of the meeting. This Society meets regularly on the first Tuesday evening of each month.

The Missouri Mining Association, at the Missouri School of Mines and Metallurgy, Rolla, Mo., has elected officers for the session 1914-15 as follows: President, W. M. Benham; Vice-President, E. A. Miller; Secretary, W. H. McCartney; Treasurer, J. C. Miller. Moving pictures showing the workings and processes of the National Lead Co. were shown at a recent meeting. The last meeting was devoted to an illustrated lecture on the Michigan Copper District.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Dec. 10, 1914, to Jan. 10, 1915:

William E. Saunders, Philadelphia, Pa.
F. H. Winsor, Jr., Salt Lake City, Utah.
Cecil W. Pocock, Peru, S. A.
Willet G. Miller, Toronto, Canada.

H. J. Brown, Chester, Pa.
H. Mortimer-Lamb, Montreal, Canada.
B. W. Vallat, Detroit, Mich.
W. D. B. Motter, Jr., Benson Mines, N. Y.

Adolphe E. Borie has resigned as Chairman of the Board of Directors of the Taylor-Wharton Iron & Steel Co., to devote his time to the American Road Machinery Co. and the Good Roads Machinery Co., as Chairman of the Board of Directors.

Andrew C. Lawson has been appointed Dean of the School of Mining of the University of California in place of the late Samuel B. Christy.

F. F. Amsden has been appointed Furnace Manager by J. E. Thropp, of Everett, Pa.

W. McA. Johnson has resigned as First Vice-President of the Johnson Electric Smelting, Inc., but continues as Director and Consulting Engineer of the company.

Henry J. Freyn has resigned as Third Vice-President of the H. Koppers Co., of Chicago, Ill.

J. H. Polhemus, manager of mines for the American Zinc, Lead & Smelting Co., has resigned to become Assistant General Manager of Mines for the New Jersey Zinc Co., 55 Wall Street, New York, N. Y., succeeding **W. A. Pomeroy**.

A. H. Boyd, mining and mechanical engineer, formerly with the Denver Rock Drill Manufacturing Co., has associated himself with the C. H. Shaw Pneumatic Tool Co., of Denver, Colo.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, five years' experience as mining engineer and chemist. Actual experience in underground work, surveying, mapping, foreman, etc. Will accept very moderate salary to start with reliable company. Willing to go anywhere. No. 184.

Member, technical graduate, with experience as engineer and metallurgist for various copper companies, is open for similar positions. No. 185.

Member, specializing in cyanide treatment and cyanide plant construction, with 10 years' experience in Mexico and in the West, desires position. No. 186.

Member, technical graduate, first in mining engineering and then in chemistry and metallurgy, Lafayette College; Fellow, School of Mines, Columbia University; special work Royal School of Mines, Freiberg, Saxony, Germany; ten years' experience teaching mining, chemistry, assaying, and metallurgy, desires position as Director of School of Mines or Professor of Metallurgy. No. 187.

THE ENGINEERING FOUNDATION

A noteworthy incident in the history of the Profession of Engineering in the United States was the inauguration of The Engineering Foundation on Jan. 27, 1915, in the auditorium of the United Engineering Society in New York.

The Engineering Foundation is the name given to a fund to be administered for the advancement of the Arts and Sciences connected with Engineering and the benefit of Mankind, the basis of which is the initial gift of a considerable sum by a noted engineer for this purpose. The American Society of Civil Engineers, the American Institute of Mining Engineers, the American Society of Mechanical Engineers, and the American Institute of Electrical Engineers, are to be represented equally in the administrative Board of The Engineering Foundation by election by the Board of Trustees of United Engineering Society, which has been made the custodian of the fund.

BOSTON LOCAL SECTION*Executive Committee***HENRY L. SMYTH, Chairman****ALFRED C. LANE, Vice-Chairman****AUGUSTUS H. EUSTIS, Secretary-Treasurer, 131 State St., Boston, Mass.****ROBERT H. RICHARDS****ALBERT SAUVEUR****Meets on the first Monday evening of each winter month***Twenty-fourth Meeting*

The Twenty-fourth meeting of the Boston Local Section was held at the Exeter Street Theater, Boston, Monday, Jan. 4, 1915.

The Chairman, Henry L. Smyth, introduced Mr. S. Le Fevre, of Mineville, N. Y., who presented an exhibit of moving pictures, illustrating the various operations of mining and milling iron ore as carried out at Mineville. Mr. Le Fevre pointed out, as the pictures were run off, the dangers of the various operations, and the precautions that should be taken to avoid or minimize these dangers.

After the exhibition of pictures, the members adjourned to the University Club, where dinner was served.

At the close of the dinner and an impromptu recital by Mr. Weeks, the meeting was adjourned.

AUGUSTUS H. EUSTIS, Secretary.

PUGET SOUND LOCAL SECTION*Executive Committee***I. F. LAUCKS, Chairman****J. F. MENZIES, Vice-Chairman****GLENVILLE A. COLLINS, Secretary-Treasurer, Box 144, Seattle, Wash.****W. C. BUTLER****H. L. MANLEY****Meets on second Saturday of each month**

The Puget Sound Local Section of the Institute has started a series of public meetings, at which addresses on technical subjects of public interest are to be presented.

The first of these meetings took place Dec. 14, 1914, and was attended by more than 600 persons. I. F. Laucks, Chairman of the Section, presided and delivered the introductory address, after which Dr. Charles E. Weaver, Professor of Geology at the State University of Washington, delivered an illustrated lecture on Oil Fields in Washington, and gave a very comprehensive analysis of the subject.

The Puget Sound Section has planned to hold two more such meetings during the Winter.

GLENVILLE A. COLLINS, Secretary.

VOLUME X

Owing to the scarcity which has existed for some time in the available copies of Volume X of the *Transactions*, this volume was reproduced by order of the Board of Directors. There are many members of the Institute who, no doubt, lack this volume in their complete sets and we therefore call the attention of all members to the present availability of this volume, which will be sold at \$5 in paper covers, and \$6 in half morocco.

In this connection it is to be noted that Volume IX is now out of stock. In case a sufficient number of orders for it are received to justify reproduction, a new issue of Volume IX will be made. Owing to the very large sale of *Transactions* in response to the recent special offer, authorized by the Board of Directors, a great many volumes have been sold and other volumes than IX and X are now becoming rare, so that the reduced prices cannot be continued longer. The opportunity now offered to members to secure complete sets of the *Transactions* for their libraries at very reasonable prices must be grasped in the near future. Another circular will be sent out in a short time, advertising *Transactions* at a somewhat higher price, and the price may be expected to rise from this time forth.

RESEARCH FELLOWSHIPS IN THE ENGINEERING EXPERIMENT STATION, UNIVERSITY OF ILLINOIS

To extend and strengthen the field of its graduate work in engineering, the University of Illinois has since 1907 maintained ten Research Fellowships in the Engineering Experiment Station. These fellowships, for each of which there is an annual stipend of \$500, are open to graduates of approved American and foreign universities and technical schools. Appointments to these fellowships are made and must be accepted for two consecutive collegiate years, at the expiration of which period, if all requirements have been met, the Master's degree will be granted. Not more than half of the time of the Research Fellows is required in connection with the work of the department to which they are assigned, the remainder of the time being available for graduate study.

Nominations to fellowships, accompanied by assignments to special departments of the Engineering Experiment Station, are made from applications received by the Director of the Station each year not later than the first day of February. These nominations are made within the month of February by the Station Staff, subject to the approval of the Faculty of the Graduate School and the President of the University. Appointments are made in March, and they take effect the first day of the following September. Vacancies may be filled by similar nominations and appointments at other times.

Nominations to these fellowships are based upon the character, scholastic attainments, and promise of success in the principal line of study or research to which the candidate proposes to devote himself. Preference is given those applicants who have had some practical engineering experience following their undergraduate work.

The Engineering Experiment Station, an organization within the College of Engineering, was established in 1903 for the purpose of carrying on investigations in the various branches of engineering, and for the study of problems of importance to engineers and to the manufacturing and industrial interests of the State. Research work may be undertaken in architecture, architectural engineering, chemistry, civil engineering, electrical engineering, mechanical engineering, mining engineering, municipal and sanitary engineering, physics, railway engineering, and in theoretical and applied mechanics.

The work of the Station is closely related to that of the College of Engineering, and the Heads of Departments in the College constitute the administrative Station staff. Investigations are carried on by the members of the staff and other members of the instructional force of the College of Engineering, by special investigators employed by the Station, and by the Research Fellows.

Additional information may be obtained by addressing: The Director, Engineering Experiment Station, University of Illinois, Urbana, Ill.

REGISTRATION AT PANAMA-PACIFIC EXPOSITION

Professor Van Barneveld, director of the Palace of Mines of the Panama-Pacific Exposition, has kindly consented to keep in his office a register of mining engineers and metallurgists who may visit the Exposition and who will report to his office their names and local addresses. Through this courtesy on the part of Professor Van Barneveld, members of the Institute may get in touch with other mining engineers and metallurgists who may be visiting the Exposition at the same time as themselves.

INSPECTING FIRE HAZARDS

The Committee on Field Practice of the National Fire Protection Association has completed its two years' work in the compilation of an inspection manual. This publication is called Field Practice to distinguish it from an ordinary fire protection handbook, from which it differs radically in function. It is not a mere compilation of fire protection standards, but a handbook designed to educate and serve the man who is undertaking inspection work, and who, possibly, has had very little previous experience. The increasing inspection of premises by uniformed members of the fire departments and by newly constituted municipal inspection bureaus has made such a handbook imperative, covering not only standard equipments, but covering what may be called points of relaxation from the standard which the inexperienced inspector does not know how to look for. This book is designed to point out the common faults in equipments and those points of deterioration difficult for inexperienced persons to discover, with methods and suggestions for their remedy. It is, in its potential usefulness, the most important publication which the National Fire Protection Association has ever compiled.

The Association has published a list of its pamphlets, standards, bulletins, fire reports, etc., which will be sent on application to the Secretary, 87 Milk Street, Boston, Mass.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays; from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

LEHRBUCH DES TIEFBAUES. Ed. 5. Bd. 1. By Esselborn. Leipzig, 1914.

DER ELEKTRISCHE STARKSTROM IM BERG UND HÜTTENWESEN. By W. von Winkler. Stuttgart, 1905.

MINING AND QUARRY INDUSTRY OF NEW YORK STATE. Bull. 174, N. Y. State Museum. Albany, 1914.

PRACTICAL STAMP MILLING AND AMALGAMATION. Ed. 3. By H. W. MacFarren. Mining and Scientific Press, San Francisco, 1914. (Gift of Publishers.)

[NOTE.—This is the third edition of this useful manual, first published in 1910, which has been highly appreciated by practical mill men ever since. The book has been entirely rewritten and reprinted, with a larger page, and numerous illustrations. A final chapter on the Arrangements and Construction Costs of Stamp Mills, by Charles T. Hutchinson, formerly manager of the mining machinery departments, first of the Union Iron Works, and later of the Joshua Hendy Iron Works (both in

San Francisco), adds to the value of the work, which reflects credit upon Mr. MacFarren, whose wide experience and critical judgment are impressed upon its pages. —R. W. R.]

- WERKZEUGSTAHL. Ed. 2. By Otto Thallner. Freiberg in Sachsen, 1904.
 L'ART DE TREMPER LES FERS ET LES ACIERS. By M. Camus. Paris, 1846.
 CALCULATION OF BLAST-FURNACE CHARGES. (In Russian.) By M. A. Pavloff. St. Petersburg, 1914. (Gift of Author.)
 DAS EISENHÜTTENWESEN. Ed. 4. By F. W. Wedding. Leipzig, 1912.
 DIE EISENINDUSTRIE. By Oskar Simmersbach. Leipzig, 1906.
 ETUDES THÉORIQUES ET PRATIQUES SUR LES PROPRIÉTÉS ET L'EMPLOI DE L'ACIER. By J. B. J. Dessoye. Paris, n. d.
 GIESSEREISEISEN UND GUSSWAREN. Ed. 2. By Ad. Vieth. Bremen, 1911.
 TIN PLATE INDUSTRY. By J. H. Jones. London, 1914.
 DAS ÄTZEN UND FÄRBen DER METALLE. By Georg Buchner. Berlin, 1911.
 SCHMELZEREI, GIESSEREI UND PUTZEREI. Ed. 2. By Ad. Vieth. Bremen, 1912.
 MÉMOIRE SUR LA FABRICATION ET LE COMMERCE DES FERS À ACIER DANS LE NORD DE L'EUROPE. By M. F. LePlay. Paris, 1846.
 DIE METALLE. By Bernhard Neumann. Halle a. S., 1904.
 MÉTALLURGIE DU FER EN ANGLETERRE. By M. Gruner et M. Lan. Paris, 1862.
 DAS ROHEISEN. Ed. 4. By A. Ledebur. Leipzig, 1904.
 SIDEROLOGY, THE SCIENCE OF IRON. By Hanns Freiherr v. Jüptner, translated from the German by Charles Salter. London, 1902.
 UNTERSUCHUNGEN ÜBER WALDRUCK UND KRAFTBEDARF BEIM AUSWALZEN VON KNÜPFELN, WINKELN. By J. Puppe. Düsseldorf, 1913.

Geology, Mineralogy, and Mineral Production

- ECONOMIC GEOLOGY OF THE BELGIAN CONGO, CENTRAL AFRICA. By S. H. Ball and M. K. Shaler. (Reprinted from *Economic Geology*, Oct., 1914.) N. p., 1914. (Gift of S. H. Ball.)
 NEUES JAHRBUCH FÜR MINERALOGIE, GEOLOGIE UND PALÄONTOLOGIE. Beilageband 38, pt. 2. Stuttgart, 1914.
 TEXT-BOOK OF GEOLOGY. By James Park. London, 1914.
 CONTRIBUTIONS TO ECONOMIC GEOLOGY, 1912. Part II—Mineral Fuels. Bull. 541, U. S. Geological Survey. Washington, 1914.
 METALLBANK UND METALLURGISCHE GESELLSCHAFT. Comparative Statistics of Lead, Copper, Spelter, Tin, Aluminum, Nickel, Quicksilver and Silver. 20th Annual Issue, 1904-1913. Frankfort on the Main, 1914. (Gift of Metallbank und Metallurgische Gesellschaft.)
 MINERAL PRODUCTION FOR 1913. Bull. 68, California State Mining Bureau. San Francisco, 1914.
 PRODUCTION OF COAL AND COKE IN CANADA, 1913. Ottawa, 1914.

Chemistry and Physics

- HANDBUCH DER MINERALCHEMIE. Band 11, pt. 6. By C. Doelter. Dresden, 1914.
 LABORATORIUMBUCH FÜR DEN EISENHÜTTENCHEMIKER. By Max Orthey. Halle a. S., 1907.

Non-Metallic Minerals

- COAL MINING PRACTICE IN DISTRICT VI. Bull. 8, Illinois Coal Mining Investigations. Urbana, 1914.
 DIE ASBESTZEMENTSCHIEFER-FABRIKATION. By K. A. Weniger. Berlin, 1914.
 GYPSUM IN CANADA. By L. H. Cole. Ottawa, 1913.
 HANDBOOK ON PETROLEUM FOR INSPECTORS UNDER THE PETROLEUM ACTS. Ed. 3. By J. H. Thomson and Boverton Redwood, revised by A. Cooper-Key. London, 1913.
 LES PIERRES PRÉCIEUSES. By Jean Escard. Paris, 1914.
 NOTES ON POTTERY CLAYS. By James Fairie. London, 1901.
 OIL CONQUEST OF THE WORLD. By F. A. Talbot. Philadelphia, 1914.
 SCHWARZPULVER UND SPRENGSALPETER. Ed. 2. By Richard Escapes. Leipzig, 1914.

General

DICTIONARY OF SPANISH, SPANISH-AMERICAN, PORTUGUESE, AND PORTUGUESE-AMERICAN MINING, METALLURGICAL AND ALLIED TERMS. Ed. 2. By Edward Halse. London, 1914.

REPORT OF THE SHELBY SMELTER COMMISSION. N. d.

Company Reports

LA MINA DE "LA ESTRELLA" EN CUALE. By D. V. Navarro. Guadalajara, 1908. (Gift of Kirby Thomas.)

Trade Catalogues

DOBBINS CORE DRILL Co., New York, N. Y. The Drill with 25 to 50 per cent. plus efficiency.

GENERAL ELECTRIC Co., Schenectady, N. Y.

Bulletin 42010. Small turbo-generator sets. 1914.

Bulletin 42300. Steam engine-driven generating sets. Nov., 1914.

RIDGWAY'S HINGE-EDGE BELT CONVEYOR. Circular.

SANFORD-DAY IRON WORKS, Knoxville, Tenn. "Whitney" roller-bearing trucks.

SUPPLEE-BIDDLE HARDWARE Co., Philadelphia, Pa. Monel Metal. Dec., 1914.

VOLAND & SONS, New Rochelle, N. Y. Short Beam Analytical Balance, No. 1006-C.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Dec. 10, 1914, to Jan. 10, 1915:

Members

- AGUILAR-REVOREDO, J. F., Min. Engr.....Casilla 146, Oruro, Bolivia.
 BAKER, JOHN M., Min. Engr.....Baker Mines Co., Cornucopia, Ore.
 BETTS, ROBERT M., Mgr.....Cornucopia Mines Co., Cornucopia, Ore.
 BILLINGS, THOMAS PARRY, Min. Engr., 404 Dooley Block, Salt Lake City, Utah.
 BLAIR, GEORGE SHEPPARD, Engr. and Assayer, Mariposa Min. & Mill. Co.,
 Hornitos, Cal.
 BOLE, JOHN W., Supt...Siskiyou Dredging Co., Fort Jones, Siskiyou Co., Cal.
 BOWMAN, REGINALD G., Statistician, International Smelting Co., 621 Kearns Bldg.,
 Salt Lake City, Utah.
 BRIBACH, OSCAR N., Chem.Atlas Min. & Mill. Co., Ouray, Colo.
 BRUCE, ALBERT EDWIN, Met.....W. R. Grace & Co., La Paz, Bolivia.
 BURNETT, E. V., Min. Engr., Société Française des Mines de Cuivre de Collahuasi La
 Grande, via Antofagasta, Chile.
 CARPENTER, EDWIN LEON, Mine Supt.Black Hawk Coal Co., Black Hawk, Utah.
 CARPENTER, JAMES R., Chief Engr., Consolidated Fuel Co., Black Hawk,
 via Price, Utah.
 CARSON, JOSEPH A., Sales Engr...Care Allis-Chalmers Mfg. Co., Milwaukee, Wis.
 CHADWICK, JOHN P., Chief Chemist.....Braden Copper Co., Rancagua, Chile.
 CUNNINGHAM, BURTON LEIGH, Asst. Geol...Southern Pacific Co., Palo Alto, Cal.
 DAY, HARRY L., Mine Mgr.....Federal Mining & Smelting Co., Wallace, Ida.
 DVORKOVITZ, PAUL, Petroleum Expert, 4 Broad Street Place, London, E. C., England.
 ELY, FRED B., Field Engr.....Gum-Thompson Co., Superior, Ariz.
 EVANS, GEORGE D., Engr. and Contractor, 317 Thompson Bldg., Pottsville, Pa.
 FEARING, FREDERICK C., Petroleum and Oil Products, 1416 S. Penn Square,
 Philadelphia, Pa.
 GALLACHER, JAMES.....Genl. Supt. Mines, 1302 N. 32d St., Birmingham, Ala.
 GEIGER, WILLIAM FREDERICK, Experimental Work, Detroit Copper Co.,
 Morenci, Ariz.
 GIBBS, CHARLES H., Geol.....Utah Fuel Co., Salt Lake City, Utah.
 GOODRICH, HARRY CHINTON, Chief Engr., Utah Copper Co., McCormick Bldg.,
 Salt Lake City, Utah.
 GOW, PAUL A., Supt.....Pilot-Butte Mining Co., Butte, Mont.
 HANFORD, JABEZ B., Genl. Supt., Elkins Coal & Coke Co., Morgantown, W. Va.
 HAZLEWOOD, STUART, Sales Engr., Midvale Steel Co., 459 Monadnock Bldg.,
 San Francisco, Cal.
 HICE, RICHARD R., State Geol.....Beaver, Pa.
 HIXON, HAROLD GARFIELD, Met....Supt., Prime Western Spelter Co., Iola, Kan.
 HOPKINS, EDWARD W., Mine Mgr.....Commonwealth, Wis.
 ISEMAN, PERCY R.....505 Fifth Ave., New York, N. Y.
 JONES, JAMES ELLWOOD, Min. Engr.....Switchback, W. Va.
 JORDAN, KNIGHT STARR, Mgr.....Knight-Christensen Met. Co., Provo, Utah.
 KAERCHER, EDWARD EDGAR, Genl. Supt., Philadelphia & Reading Coal & Iron Co.,
 Pottsville, Pa.
 KNEASS, STRICKLAND, JR., Asst. Steam Engr., Youngstown Sheet & Tube Co.,
 Youngstown, Ohio.
 LEACH, ALBERT ALLYN, JR., Mgr., Sonora Exploration & Metals Co.,
 505 W. Marble St., Albuquerque, N. M.
 LEWYN, OSWALD MARK, Supt. Mines.....Braden Copper Co., Rancagua, Chile.
 MACGREGOR, ALEXANDER GRANT, Engr.....Globe, Ariz.
 MCCROSSIN, E. FRANCIS, Min. Engr.....1824 S. 12th Ave., Birmingham, Ala.
 MCGOVERN, R. A., Min. Engr., Care Caloric Co., 17 Battery Place, New York, N. Y.
 MILLER, CHARLES LEO, Contracting Engr.....1511 Virginia St., Charleston, W. Va.
 MORGAN, HARRY J.....1604 Union St., San Francisco, Cal.
 NEWMAN, WILLIAM EDWARD, Supt., St. Louis Smelting & Refining Co.,
 Collinsville, Ill.

NEWTON, EDMUND, Met., Mines Experiment Station, Minnesota School of Mines,	Minneapolis, Minn.
OSBORNE, CLARENCE BRISTOL, Chief Geol., California Highway Commission,	Sacramento, Cal.
POLLAK, ROBERT R., Petroleum Engr.....	518 Mills Bldg., San Francisco, Cal.
POLSON, ALEXANDER, Lumberman and Miner.....	Hoquiam, Wash.
REED, HUBBARD W., Mining.....	233 Atlas Block, Salt Lake City, Utah.
RODEGERDTS, C. A., Mine Valuation, Care A. A. Rosenshine, Mills Bldg.,	San Francisco, Cal.
RUMSEY, SPENCER S., Chief Engr.....	Oliver Iron Mining Co., Duluth, Minn.
SALINAS, LEOPOLDO SALAZAR, Geol.....	Dos Estrellas Mining Co., El Oro, Mexico.
SEYMOUR, R. D., Sales Agent, American Steel & Wire Co.,	1610 Walker Bank Bldg., Salt Lake City, Utah.
SHELDON, GEORGE R., Min. Engr.....	Border, Wyo.
SIELAFF, GUSTAV JULIUS, Min. Engr.....	Abangarez, Costa Rica.
SMITH, HUMPHREY D., Asst. Mgr.....	Crozer Coal & Coke Co., Elkhorn, W. Va.
SNIDER, EARL BAILLIE, Civ. and Min. Engr.....	Charleston, W. Va.
STIMPSON, CHARLES WARD, Mgr., Stimpson Equipment Co., 309 Felt Bldg.,	Salt Lake City, Utah.
STRANAHAN, C. B., Mining.....	82 Beaver St., New York, N. Y.
SWENSEN, KARL P., Mech. and Min. Engr., 6 Takiyamacho, Kyobashiku,	Tokyo, Japan.
THUM, WILLIAM, Supt.....	U. S. Metals Refining Co., East Chicago, Ind.
TURNER, THOMAS NORTON, Min. Engr.....	Press Club, San Francisco, Cal.
WAKABATASHI, YAICHIRO, Min. Engr. and Mgr., Fukiyamachi, Kawakamikori,	Okayamaken, Japan.
WOOLF, WALLACE G., Met. Research.....	Univ. of Utah, Salt Lake City, Utah.
WRATHEE, WILLIAM EMBRY, Oil Geol.....	Wichita Falls, Tex.
WREN, ANDREW ADDISON, Min. Engr.....	McGill, Nev.
WROTH, JAMES S., Min. Engr..	Care Chile Exploration Co., Chuquicamata, Chile.

Associate Members

BROOKER, CHARLES FREDERICK, Mfr.....	Ansonia, Conn
LAMONT, THOMAS WILLIAM, Banker.....	23 Wall Street, New York, N. Y.
MCCARRICK, E., Mining.....	628 N. Serrano Ave., Los Angeles, Cal.

Junior Members

ANDERSON, RAY B., Student.....	P. O. Box 578, Golden, Colo.
GOLDSTEIN, DAVID E., Min. Engr.....	157 Jefferson St., Passaic, N. J.
GRAHAM, WALTER FRANKLIN, Met. Engr.....	1828 E. 65th St., Cleveland, Ohio.
JOHNSTON, FRANCIS B., Apprentice, Carnegie Steel Co., Carnegie Hotel, Munhall, Pa.	
PITTMAN, FRANK L., Millman.....	Butte Duluth Mining Co., Butte, Mont.
WELLS, RALPH EVANS, JR., Asst. Chem., American Smelting & Refining Co.,	Murray, Utah.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Dec. 10, 1914, to Jan. 10, 1915, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Louis Winfield Adams, Bethlehem, Pa.

Proposed by E. O'C. Acker, C. A. Buck, W. L. Cummings:

Born 1880, Washington, D. C. 1899, Washington High School. 1903, Mass. Inst. of Tech., Mechanical Dept.; S. B. 1903, Illinois Steel Co. 1904, Lackawanna Steel Co. 1906, Illinois Steel Co. 1909, Asst. Supt., Bethlehem Steel Co.

Present position: Supt., Rolling Mills, Lehigh Plant, Bethlehem Steel Co.

Harry Mayo Andreen, Treadwell, Alaska.

Proposed by J. C. Branner, C. F. Tolman, Jr., D. M. Folsom.

Born 1891, Woodside, Cal. 1909-13, Stanford Univ.; A. B. 1913, Assayer, Jualin Mine, Jualin, Alaska. 1913-14, Asst. Assayer, Alaska Treadwell Gold Mining Co., Treadwell, Alaska.

Present position: Millman, Cyanide Plant, Alaska Treadwell Gold Mining Co.

Lester Armond Blackner, Ray, Ariz.

Proposed by L. S. Cates, W. S. Boyd, Robert H. Bradford.

Born 1886, Beaver, Utah. 1902, Grad. Public School, Salt Lake City, Utah. 1902-06, Salt Lake High School. 1906-10, Univ. of Utah, Salt Lake City, Utah. 1906, Draftsman, Mine & Smelter Supply Co., Salt Lake City, Utah. 1907-08, Engineering Dept., Boston Cons. Mining Co., Bingham, Utah. 1908, Architectural Dept., Oregon Short Line R. R., Salt Lake City, Utah. 1908-09, Utah Mines Coalition, Silver Lake (Brighton), Utah.

Present position: 1910 to date, Stope Engr., Ray Cons. Copper Co.

James Bunten, Canon City, Colo.

Proposed by G. H. Cox, C. R. Forbes, H. A. Buehler.

Born 1886, Coal Creek, Colo. 1902-05, Canon City High School. 1906-10, Missouri School of Mines; B. S. 1909, Summer, Empire Zinc Co., Canon City, Colo. 1910, Empire Zinc Co.

Present position: In business, Bunten & Curtis, Civil and Mining Engineers.

Harry Edward Bush, Keddie, Cal.

Proposed by S. E. Bretherton, Frank L. Sizer, Charles V. Weston.

Born 1880, Redding, Cal. Belmont School, Cal. (Preparatory). 1905, Stanford Univ. 1905-07, Assayer and Surveyor, Afterthought Copper Co. 1907-10, Field Engineer, later in charge of mine, Balaklala Cons. Copper Co. 1911, Personal field work. 1912-13, Engineering and geology, Nevada Douglas Copper Co.

Present position: Supt., Engels Copper Mining Co.

John Winthrop Chandler, Tonopah, Nev.

Proposed by Horace V. Winchell, Walter H. Wiley, Warren V. Richardson.

Born 1877, Wellington, Kan. 1901, Grad., State School of Mines, Colo.; E. M. 1901-02, Supt., Midas Gold Min. Co., Deep Creek, Utah. 1902-04, Field work; headquarters, San Francisco, Cal. 1904-09, Supt., McNamara Min. Co., Tonopah, Nev. 1909-12, Exam. work, San Francisco, Cal. 1913, Supt., Tonopah North Star Min. Co. and Rescue-Eula Min. Co., Tonopah, Nev.

Present position: Genl. Supt., West End Cons. Min. Co.; Supt., Halifax Tonopah Min. Co.

George W. Coggeshall, Washington, D. C.

Proposed by E. W. Parker, Charles E. Munroe, Anthony F. Lucas.

Born 1867, Des Moines, Iowa. 1890, Iowa College, Grinnell; B. C. 1891-92, Harvard Graduate School, Chemistry. 1892-95, Leipzig, Germany, Chemistry, Math., Phys.; Ph. D. 1889-90, Student, Asst. Chem., Iowa College. 1892, Instr. Chem., Harvard Summer School. 1895-97, Instr. Phys., Chem., Harvard Graduate School. 1898-1911, Mgr., later Treas. and Pres., Eastern Chemical Co., Boston, Mass.—Dr. A. S. Cushman and the writer founded this company in 1895.

Present position: 1911 to date, Member of the Institute of Industrial Research, Washington, D. C., and in charge Division of Mill Problems.

John E. Daley, Clarksville, Ark.

Proposed by H. Denman, Franklin Bache, Walter Gilman.

Born 1867, Pittston, Pa. 1884-92, Ordinary positions in and around coal mines. 1892-96, Mine foreman, Butler Coal Co., Pittston, Pa. 1896-1904, Genl. foreman at group of mines (5) for Erie Coal Co., Pittston, Pa. 1904, Genl. Supt. Consolidated Coal Co., Spadia, Johnson Co., Ark. 1905-11, Genl. Mgr., Scranton Anthracite Coal Co., Montana, Johnson Co., Ark. 1911-14, Individual coal-mine operator in Spadia and Montana, Johnson Co., Ark.; Hackett, Sebastian Co., Ark. During above periods, made expert examination and reports to Chancery Courts, Federal Courts, individuals and corporations. Settlements in Chancery Courts were all on basis of my reports.

Present position: Pres. and Genl. Mgr., Eustice, Daley & Eustice Coal Co.

Roy Henry Davis, Gary, W. Va.

Proposed by Howard N. Eavenson, E. O'Toole, H. W. Saunders.

Born 1885, Amity, Ore. 1900, Public School, Eugene, Ore. 1904, High School, Eugene, Ore. 1904, Four months, Univ. of Oregon, Eugene, Ore. 1905-09, U. S. Naval Academy, Annapolis, Md. 1911-13, Post-graduate course in Naval Ordnance.

Present position: 1913 to date, Mech. Engr., U. S. Coal & Coke Co.

Charles Alexander Filteau, Brooklyn, N. Y.

Proposed by J. V. N. Dorr, H. N. Spicer, Burr A. Robinson.

Born 1879, Idaho Springs, Colo. 1907, Colo. School of Mines; M. E. 1904-05, Sun & Moon M. & M. Co., Idaho Springs, Colo. 1906-07, Filteau & Arkell's Assayers and Chemists, Idaho Springs. 1907-08, Lundberg, Dorr, Wilson & Mogul M. Co., Terry and Pluma, S. D. 1908-09, Indé Gold Mining Co., Indé, Durango, Mexico. 1909-10, San Jose de Gracia, Sinaloa, Mexico. 1911-14, Colo. Mining Co., Manila, P. I.

Present position: Genl. Mgr., Inch Mining & Development Co., San Domingo, Peru.

Douglas Warner Franchot, Tulsa, Okla.

Proposed by Anthony F. Lucas, William J. Linn, Frank Peterson.

Born 1880, Olean, N. Y. 1899-1903, Yale, Sheffield Scientific School; Ph. D. 1903-14, Producer of crude petroleum in various oil fields of U. S., owner and supt. Mfr. of liquefied natural gas gasoline by a process perfected by myself.

Present position: Owner and Mgr. of D. W. Franchot & Co.

Philip L. Gill, Woodmere, Long Island, N. Y.

Proposed by James Douglas, J. B. F. Herreshoff, James B. Herreshoff, Jr.

Born 1884, Brooklyn, N. Y. 1894-1901, Brooklyn Polytechnic Institute. 1901-05, Princeton Univ.; A. B., Specialized in chemistry and civil engng. 1905-10, Asst. Chemist and Met., Nichols Copper Co., experience in smelting, converting, reverberatory furnace and electrolytic refining. 1910-14, Asst. Supt. Wire Bar Dept., Nichols Copper Co., Laurel Hill Wks.

Present position: 1914 to date, President, General Ore Concentrating Co.

Charles Denham Grier, Blair, Nev.

Proposed by Lyon Smith, Charles W. Merrill, Frank B. Miller.

Born 1889, Denver, Colo. 1908, Manual Training High School, Denver, Colo. 1912, Colorado School of Mines, Golden, Colo.; E. M., 1912, Solution man, Northern Nevada Mines Development Co.

Present position: 1912 to date, Pittsburg Silver Peak Gold Mining Co.

William T. Griswold, Pittsburgh, Pa.

Proposed by Anthony F. Lucas, E. W. Parker, M. R. Campbell.

Born 1859, Brooklyn, N. Y. 1881, School of Mines, Columbia; C. E. 1881-83, Sinaloa & Durango R. R., Mexico. 1883-87, U. S. Geol. Survey. 1907-09, Guffey & Gailey, Pittsburgh.

Present position: 1909 to date, Geol., Phila. Gas Co., Pittsburgh, Pa.

William Ranald Grunow, Morenci, Ariz.

Proposed by M. H. McLean, Frank W. McLean, F. H. Hayes.

Born 1890, New Haven, Conn. 1904-08, Diploma, High School, Bridgeport, Conn. 1908-12, Columbia Univ.; B. S. 1910-14, Columbia School of Mines; E. M. 1911, Rolling mills, American Brass Co., Waterbury, Conn. 1913, Summer, mining trip in connection with mining course at Morenci, Ariz., with the Detroit Copper Mining Co.

Present position: Sampler, Detroit Copper Mining Co.

Edmond A. Guggenheim, New York, N. Y.

Proposed by Judd Stewart, Willard S. Morse, P. A. Mosman.

Born 1888, St. Gall, Switzerland. Dr. Sach's Collegiate Inst. 1908, Columbia University. 1911, Yale Scientific School. 1911, Amer. Smelt. & Ref. Co., Baltimore, Md. 1912, Amer. Smelt. & Ref. Co., Perth Amboy, N. J.

Present position: 1914 to date, M. Guggenheim's Sons; Director in Chile Copper and Braden Copper Co.

John Edward Hackford, Minatitlan, V. C., Mexico.

Proposed by Frank C. Laurie, Thomas J. Ryder, Anthony F. Lucas.

Born 1885, Dudley, Warwickshire, England. 1902-10, Univ. College, Nottingham, England. 1910, B. honors, London. Fellow of the Institute of Chemistry of Great Britain and Ireland (A. I. C. in 1911). Fellow of the Chemical Society. 1910-11, Chemist, Russian Steam Oil Co., Petrograd, Russia. 1911-12, Chemist, W. Ropes & Co., Petroleum Oil Refiners, Petrograd, Russia.

Present position: 1912 to date, Chemist, Mexican Eagle Oil Co.

Orville M. Hand, Cornucopia, Ore.

Proposed by D. C. Bard, Bancroft Gore, C. H. Bowman.

Born 1867, Wisconsin. General mining for past 25 years. 1895-98, Supt. of Minne Ha Ha mine, Rossland, B. C. 1899, Mine Broker, Republic, Wash. 1903, Supt., Mattoon Gold Min. Co., Baker, Ore.

Present position: 1910 to date, Supt., Queen of the West Mines Co.

Arthur Howard Higgins, London, N. W., England.

Proposed by Hugh K. Picard, H. M. Ridge, S. J. Speak.

Born 1880, Leeds, England. 1894, various elementary schools. 1894-98, Bradford Technical College Day School. 1898-1901, Royal School of Mines; R. S. M. in metallurgy. 1901-02, Asst. Demonstrator in Geology and Mineralogy, Royal School of Mines.

Present position: 1903 to date, Chief Met., Minerals Separation, Ltd.

George Milton Jones, Oak Hill, Fayette Co., W. Va.

Proposed by J. B. Cunningham, C. E. Krebs, Howard N. Eavenson.

Born 1886, Oak Hill, W. Va. 1906, Grad., Lawrenceville School. 1910, Grad., Princeton Univ.; C. E.

Present position: Pres., Argyle Coal Co.; Va.-Buffalo Coal Co.; Vice-Pres., Starr Coal & Coke Co.

Frank J. Kelly, Mispah, Alberta, Canada.

Proposed by Ellsworth Daggett, Elmer H. Lawall, Rufus J. Foster.

Born 1876, San Francisco, Cal. 1893, Seattle High School. Acme Business College, Seattle. Grad., International Correspondence School of Mines, "metallmining." Studied assaying with John O'Sullivan, Vancouver, B. C., Canada; and surveying with F. M. Miller, Grass Valley, Cal. 1902-03, Mine foreman, Cornell Mine, Texada Island, Canada. 1907, Mine foreman, Nevada Co., Berlin, Nev. 1908, Mine foreman, Butter's Divisadera Mines, C. A.

Present position: 1909 to date, Supt., W. M. Johnston & Co.

F. C. Langenberg, Cambridge, Mass.

Proposed by Charles H. White, Albert Sauveur, H. M. Boylston.

Born 1890, Beverly, Ohio. 1912, Ohio Univ.; S. B. 1913, Ohio Univ.; S. M. Graduate student Harvard Mining School since 1913. 1912-13, Instructor in Elec. Engrg., Ohio Univ.

Present position: Asst. to Prof. Sauveur, Harvard Univ.

John K. MacGowan, New York, N. Y.

Proposed by P. A. Mosman, A. Eilers, H. A. Guess.

Born 1874, Philadelphia, Pa. 1881-90, Girard College, Phila., Pa. Previous to 1900, railroad business in Western States.

Present position: 1900 to date, Director and member of Executive Committee, American Smelting & Refining Co.

Edwin Bernard Nagle, Santiago, Cuba.

Proposed by Benjamin B. Lawrence, Franklin Guiterman, E. H. Emerson.

Born 1880, Winona, Minn. General education at public schools, Winona, Minn. 1899, Miner, Iroq Belt Mining Co., Iron Belt, Wis. 1903, Timberman, El Cobre Mines.

Present position: 1909 to date, Genl. Supt., Cuba Copper Co.

Eusebio Paulo de Oliveira, Rio de Janeiro, Brazil.

Proposed by Horace E. Williams, I. C. White, J. F. Kemp.

Born 1882, Abaethé, Minas Geraes. 1905, Min. Engr., School of Mines, Ouro Preto, Minas Geraes. 1905, Survey of copper mines in Bahia. 1906-07, On survey of coal regions of South Brazil with Dr. I. C. White. 1908-14, Geol., Brazilian Geological Survey. 1914, Geol., Roosevelt-Randon Expedition.

Present position: Geol., Brazilian Geological Survey.

James Pearson, Great Falls, Mont.

Proposed by Lewis Stockett, Frank Scotten, Milo W. Krejci, C. W. Goodale, J. H. Klepinger, W. F. C. Whyte, W. F. McNeill.

Born 1865, Massillon, Ohio. Started to work in coal mines at the age of 11 years. Worked in different branches of mining up to the age of 26. 1891, What Cheer Coal Co.; Foreman three years, Superintendent three years. Demonstrator, Harrison Machine Co., one year. Foreman, Sand Coulee Coal Co., Montana, two years and Superintendent three years. Manager, Cottonwood Coal Co.'s mines, Sand Coulee, Mont.; later, Manager of Cottonwood Coal Co.'s properties in Stockett, Mont., and what is known as the Cokedale property in Washington, and Genl. Mgr. of all the Great Northern coal properties in the States of Montana and Washington, and have filled this position up to the present time.

Present position: Investigation work in connection with coal properties. 1912 to date, Mining Engineer for the Great Northern Railway Co. The Cottonwood Coal Co. is a subsidiary company of the G. N. Ry. Co.

William John Priestley, Bethlehem, Pa.

Proposed by E. O'C. Acker, W. F. Roberts, W. L. Cummings.

Born 1884, North of Ireland. 1900-04, Chicopee High School, Chicopee, Mass. 1904-08, Lehigh Univ., So. Bethlehem, Pa.; M. E. in Mech. Engr. 1908-09, Saucon Plant, Bethlehem Steel Co. 1909-10, Dravo Contracting Co., Pittsburgh, Pa. 1910-11, Carpenter Steel Co., Reading, Pa.

Present position: 1911 to date, Supt., Armor Plate Dept., Bethlehem Steel Co.

Charles John Reed, San Francisco, Cal.

Proposed by S. E. Bretherton, G. H. Clevenger, W. H. Shockley.

Born 1858, Dakota City, Iowa. 1881, Univ. of Mich.; B. A. Organized American Electro-Chemical Society; 1901-03, Secretary. 1895-1910, Member of Committee on Science and Arts, Franklin Institute, Philadelphia. 1882-84, U. S. Deputy Mineral Surveyor and Engr. (Surveyor), Plutus and Freeland Mfg. Co., Idaho Springs, Colo. 1888, Supt., Jenny Incandescent Lamp Co., Indianapolis. 1888-91, Making magnetic mine surveys and in Edison's Laboratory, Orange, N. J. 1893-1904, in Philadelphia, electro-chemical research, chiefly magnetic separators in connection with the Reed-Electro and later with the Security Investment Co. 1904, Hercules Metal Reduction Co., extracting brass and other metals from residues. 1892-1914, Many inventions; took out 80 patents, several on ore concentrators.

Present position: Engaged in research work and invention.

Walter Alfred Richelsen, Houghton, Mich.

Proposed by A. H. Wohlrab, F. L. Van Orden, W. E. Hopper.

Born 1886, Chicago, Ill. 1904-07, University of Ill.; C. E. 1911-13, Michigan College of Mines; B. S. 1913, post-graduate work in geology, Michigan College of Mines. 1905-06, Chairman, Draftsman, Eng. Dept., Chicago & N. W. R. R. 1907-08, Transitman, Location and Construction, Canadian Pacific Ry. 1908, Transitman, Grand Trunk Pacific Ry. 1908-10, Transitman, Draftsman and Asst. Construction Engr., Copper River & Northwestern Ry., Alaska. 1910-11, Transitman and Bridge Engr., Colo. & Southern Ry.

Present position: Construction Engr., Isle Royale Copper Co.

Thomas J. Ryan, New York, N. Y.

Proposed by Frank L. Hess, David T. Day, E. W. Parker.

Born Illinois. International Petroleum Co. and others.

Rens Edward Schirmer, Idaho Springs, Colo.

Proposed by Ott F. Heizer, S. A. Ionides, Richard A. Parker.

Born 1886, Boston, Mass. 1900, Grammar School, Boston, Mass. 1904, High School, Boston, Mass. 1908, Mass. Inst. of Tech.; B. S. 1909, Work in Mining School, Columbia Univ. 1908, Boston Cons. Min. Co.; Member, Colo. Scientific Society; National Geographic Society.

Present position: Mgr. Tremont Mining Co., Gunnell Mine, Central City, Colo.; Argo Mining, Drainage, Transportation & Tunnel Co.

Robert Glenn Sickly, Cobalt, Ont., Canada.

Proposed by Robert E. Dye, G. H. Cox, C. R. Forbes.

Born 1888, La Harpe, Kan. 1905-07, Iola High School, Iola, Kan. 1909-13, Missouri School of Mines, Rolla, Mo.; B. S. in metallurgy and chemistry. 1906-07, Asst. Chemist at zinc smelter, American Metal Co., La Harpe, Kan. 1907-09, Ore sampler, L. Vogelstein & Co., Missouri, Kansas, and Oklahoma, zinc smelters.

Present position: 1913 to date, Chemist, Buffalo Mines, Ltd.

Francis J. Strachan, Cananea, Son., Mexico.

Proposed by L. D. Ricketts, Julius Bergman, David Cole.

Born 1868, Canada. 1892-95, Collegiate Institute, Winnipeg, Manitoba. 1895-1900, Regina Gold Mining Co., Ltd., London Mines, near Kenora, Canada. 1900-03, General milling experience, Smuggler Mining Co., Aspen, Colo.

Present position: 1903 to date, Cananea Consolidated Copper Co.

Thomas Henry Thompson, Punxsutawney, Pa.

Proposed by Thomas A. Furniss, John McLeary, H. M. Kanarr.

Born 1868, Natrona, Pa. 1885, School at Natrona. 1886, High School, Shire Oaks. 1901, Complete course of Coal Mining, International Correspondence School, Scranton, Pa. 1900-13, Pittsburgh Coal Co., Monongahela Coal & Coke Co., Pittsburgh Buffalo Co.

Present position: 1913 to date, State Mine Inspector.

Timothy D. Walsh, Denver, Colo.

Proposed by F. A. Ross, L. K. Armstrong, Fred H. Bostwick.

Born, 1886, Denver, Colo. 1905, Grad., West Side High School, Denver, Colo. 1905-06, Attended Colorado School of Mines; full credits received for year's work. 1906-09, Colorado College; B. S. in Min. Engrg. 1909-10, Reporter, R. G. Dun & Co., Denver office. 1910-11, In Engrg. Dept. of Mexico Northwestern R. R., Chihuahua, Mexico, during construction, Chairman to Resident Engineer. 1911-13, Shift Boss, cyanide mill, Dolores Mines Co., Madera, Chihuahua, Mexico. 1913, Shift Boss, stamp mill, Bully Boy Co., Marysvale, Utah. 1914, Shift Boss, cyanide mill, Junta Cons. M. & M. Co., Telluride, Colo.

Present position: 1914 to date, Examining mining properties, under H. E. Crawford, New York, N. Y.

Alfred Gary White, Washington, D. C.

Proposed by D. A. Lyon, Robert H. Bradford, Robert S. Lewis.

Born 1886, Brandon, Wis. 1903-07, Lawrence College, Wis.; A. B. 1908-09, Univ. of Wis.; A. M. 1909-13, Univ. of Penn. (Wharton School), graduate work in Industrial Economics. 1909-13, Instructor in Industry, Wharton School of Finance and Commerce, Univ. of Penna.

Present position: 1913 to date, Mine Economist, Bureau of Mines.

Sidney Wilmot Winslow, Boston, Mass.

Proposed by Sidney J. Jennings, F. Y. Robertson, Willard S. Morse, F. F. Colcord.

Born 1854, Brewster, Mass. Educated in the public schools of Salem, Mass. Learned shoemaking in father's shoe factory, became much interested in shoe machinery and was actively engaged in its development for years.

Present position: 1899 to date, Pres., United Shoe Machinery Co.

Junior Members

Willard Miles Benham, Rolla, Phelps Co., Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1889, Cobden, Ill. 1904-08, Cobden High School. 1909-12, Missouri School of Mines. 1910 (summer), Rodman and construction work, Chino Copper Co. 1911-12, Miner and Assistant in Engineering Dept., Doe Run Lead Co. 1912 (summer), Mill work, Federal Lead Co. 1913 (summer), Miner, Inspiration Copper Co.

Present position: 1909 to date, Missouri School of Mines.

Maurice Lincoln Bradt, Houghton, Mich.

Proposed by F. W. McNair, F. W. Sperr, W. E. Hopper.

Born 1892, St. Charles, Mich. 1910, High School, Saginaw, Mich. 1910, Time-keeper, Wickes Boiler Co., Saginaw, Mich. 1911, San Luis Valley Sugar Co., Monte Vista, Colo. 1912, German American Sugar Co., Bay City, Mich. 1913, Saginaw Bay City R. R. Co., Saginaw, Mich.

Present position: Student, Michigan College of Mines.

Harold Osborn Davidson, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1892, Commonwealth, Wis. 1914, Univ. of Mich.; B. C. E.

Present position: Student, Univ. of Wisconsin.

Djevad Eyoub, New York, N. Y.

Proposed by Robert Peele, J. F. Kemp, Charles P. Berkey.

Born 1892, Boli, Turkey. 1905, Turkish schools in Constantinople. 1905-11, Robert College, Constantinople. 1911-12, Columbia College; B. S.

Present position: Student, Columbia School of Mines.

Ulysses S. Grant, IV, Cambridge, Mass.

Proposed by H. L. Smyth, George Sharp Raymer, L. C. Graton.

Born 1893, Westchester Co., N. Y. 1911, entered Harvard College.

Present position: Student, geological department, Harvard College.

John Edward Hanlon, Toronto, Ont., Canada.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1887, Guelph, Ont., Canada. Guelph College Institute. 1913 (summer), Nipissing mine, Cobalt, working in the melting room. 1914 (summer), Assayer, Trethewey mine, Cobalt.

Present position: Student, Univ. of Toronto.

Frederick W. Hodson, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1891, Topeka, Kan. 1899-1910, Grade and High Schools, Chicago, Ill.

Present position: Student, Univ. of Wisconsin.

Frederick Arthur Johnson, Ben Avon, Pa.

Proposed by Anthony F. Lucas, H. B. Meller, H. C. Ray.

Born 1891, New Castle, Pa. 1898-1907, Public schools, New Castle, Pa. 1907-11, New Castle High School. 1911, Univ. of Pittsburgh, School of Mines. Asst. on Engrg. Corps, Eastern Div., Penn. Lines West of Pittsburgh.

Present position: Student, Pittsburgh Univ.

Bjarne Kundsén, Madison, Wis.

Proposed by Edwin C. Holden, Richard S. McCaffery, C. K. Leith.

Born 1890, Chicago, Ill. Chemist, Youngs Mine, Iron River, Mich.

Present position: Student, Univ. of Wisconsin.

W. Harrison Loerpabel, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1892, Tenaville, S. D. 1910, Grad., Nome High School, Nome, Alaska.

Present position: Student, Univ. of Wisconsin.

Frank Leon Mills, Toronto, Ont., Canada.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1890, Brantford. Brantford High School, Harvard College Inst. 1904-10, Electrical design work, Universal Signal Co., Toronto.

Present position: Student in Mining, Univ. of Toronto.

Rollin Pallansch, Madison, Wis.

Proposed by Edwin C. Holden, C. K. Leith, Richard S. McCaffery.

Born 1891, Fredonia, Wis. 1911, Grad., Madison High School.

Present position: Student, Univ. of Wisconsin.

Leonard Lee White, Golden, Colo.

Proposed by Harry J. Wolf, William R. Chedsey, F. W. Traphagen.

Born 1888, Batesville, Ind. 1903-10, Salida High School. 1905-06, Colorado Bell Telephone Co. 1907-09, Salida, Ltd., Power & Utility Co. 1909-10, El Oro Mining & Ry. Co. 1913, Northern Colorado Power Co.

Present position: Monarch Nadoma Mining Co.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Dec. 10, 1914, to Jan. 10, 1915. This list, together with the list published in *Bulletin* Nos. 88 to 97, April, 1914, to Jan., 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of Jan. 10, 1915.

ADAMI, C. J. St. Joseph & Doe Run Lead Cos., Bonne Terre, Mo.
AMSDEN, FRANK F. Supt., Earleton Furnace, Everett, Bedford Co., Pa.

- ANDERSON, ALBERT H. Abrasive Mill Foreman, Norton Co., Worcester, Mass.
ARMAS, M. T. 48 Boulevard Emile Augier, Paris, France.
BACON, M. W. 724 Old National Bank Bldg., Spokane, Wash.
BARR, JAMES A. Engr., International Agricultural Corp., Mt. Pleasant, Tenn.
BLAUVELT, W. H., Cons. Engr., Semet-Solvay Co., 1917 West Genesee St.,
Syracuse, N. Y.
BOTSFORD, R. S. Basil Island, Bolshoi Prospect, No. 41, Kw 19, Petrograd, Russia.
BOTD, ALBERT H., C. H. Shaw Pneumatic Tool Co.,
35th and Wazee St., Denver, Colo.
BREWER, CARL Care Cleveland-Cliffs Iron Co., Ishpeming, Mich.
BROCK, R. W., Dean, Faculty of Science, Univ. of B. C., Vancouver, B. C., Canada
BROOKS, JOHN M., JR., Care Buena Tierra Mine, Santa Eulalia, Chihuahua, Mexico
BRULE, F. J. Room 1211, Hollingsworth Bldg., Los Angeles, Cal.
BURT, CURTIS F. Aurora, Nev.
CALLAWAY, F. W. Care Bunker Hill & Sullivan, Kellogg, Ida.
CAREAGA, CIPRIANO R. Larga 68, Puerto Santa Maria (Cadiz), Spain.
CARTER, EWING 321 James Bldg., Chattanooga, Tenn.
CATES, LOUIS S. Genl. Mgr., Ray Consolidated Copper Co., Ray, Ariz.
CATLETT, CHARLES, Cons. Engr. Central Bldg., Staunton, Va.
CHAPMAN, R. H. Topographic Engr., U. S. Geol. Survey, Washington, D. C.
CORNELIJSSEN, JOHANNES New Jersey Zinc Co., Franklin Furnace, N. J.
CROMWELL, ROBERT H. Hotel Morenci, Morenci, Ariz.
CUNNINGHAM, NOEL Care Copper Queen Cons. Mining Co., Douglas, Ariz.
DEL MAR, ALGERNON 1424 Alpha St., Sierra Vista, Los Angeles, Cal.
DONNER, WILLIAM H. 1703 Morris Bldg., Philadelphia, Pa.
DRAPER, CARL H. Apartado 77, Guadalajara, Jal., Mexico.
EDWARDS, R. L. Salmon, Ida.
ELKAN, BENNO. Mgr., Am. Branch, Beer, Sondheimer & Co., 61 Broadway,
New York, N. Y.
ELLIOTT, ARTHUR H. 30 Church St., New York, N. Y.
ELLIS, ERNEST W. 615 W. 3d St., Anaconda, Mont.
ELMER, WILL W. Tuttle town, Cal.
FABIAN, FRANCIS GORDON 1100 Oak Ave., Evanston, Ill.
FAIRBANK, WALLACE Cold Spring Harbour, Long Island, N. Y.
FERGUSON, CLAUDE 803 Chester Ave., Bakersfield, Cal.
FIRMSTONE, H. Longdale, Alleghany Co., Va.
FLETCHER, A. R. 781 Flood Bldg., San Francisco, Cal.
FORD, HAROLD P. 2023 S. 20th St., St. Joseph, Mo.
GARLICH, HERMAN Care St. Louis Smelting & Refining Co.,
1001-7 International Life Bldg., St. Louis, Mo.
GIBBS, GEORGE H. Stynechale Ave., Coventry, England.
GOLDSWORTHY, JOSEPH 1431 Williams Ave., N. Vancouver, B. C., Canada.
GRAPNER, M. F. General Delivery, Grass Valley, Cal.
HAFF, R. E. T. Room 329, Pennsylvania Station, New York, N. Y.
HALLOBAN, WILL Box 13, Basin, Mont.
HEISE, H. C. Care Weedon Mining Co., Notre Dame des Anges, Que., Canada.
HEROLD, STANLEY C. Care Cartagena Oil Refining Co., Cartagena, Colombia.
HIESTER, ARTHUR J. Care Empire Zinc Co., 1340 Garfield St., Denver, Colo.
HILTON, HOWARD JUDD Y. M. C. A., Phoenix, Ariz.
HINDSHAW, HENRY H. Union Springs, N. Y.
HOFFMANN, K. F. 550 Palisade Ave., Yonkers, N. Y.
HOLLISTER, SCOVILL E. Mill Supt., Compania Estanifera de Llalagua,
Llalagua, Bolivia.
HUNT, A. J. 2 Barrington Parkway, East Providence, R. I.
IHLENG, A. O., Cons. Engr. Joplin, Mo.
JAMES, WILLIAM EWART Min. Engr. and Supt., Mount Carbon Co., Ltd.,
Powellton, W. Va.
JANEWAY, J. H. 55 Wall St., New York, N. Y.
JARVIS, ROYAL P. Crested Butte, Colo.
JONES, CLEMENS AP-CATESBY Allison Bldg., Richmond, Va.
JULIAN, E. A. P. O. Box 787, Reno, Nev.
KAFFER, STEPHEN L. Care Burro Mountain Copper Co., P. O. Box 29, Tyrone, N. M.
KELLOGG, R. M. Supt., Dome Mining & Reduction Co., Bullion, Elko Co., Nev.
KEMP, L. W. Care Chile Exploration Co., Chuquicamata, via Antofagasta, Chile.
KIDDER, S. J. Mgr., Ernestine Mining Co., Mogollon, Socorro Co., N. M.
LANG, ROBERT Chickasaw, Ala.

LE CHATELIER, H. L.	Ecole des Mines, Paris, France.
LEE, CHESTER F.	1712 Hoge Bldg., Seattle, Wash.
LEE, E. C.	Box 541, Herrin, Ill.
L'ENGLE, E. FLEMING	Supt., Aguacate Mines, San Mateo, Costa Rica.
LONGYEAR, JOHN MUNRO, JR.	Leicester St., Brookline, Mass.
MACARTHUR, JOHN S.	74 York Street, Glasgow, Scotland.
MACOMB, J. DE N., JR.	Office Engr., Atchison, Topeka & Santa Fe Ry., Chicago, Ill.
MANLEY, FRANK A.	1112-3 Union Pacific Bldg., Omaha, Neb.
MARTIN, ROBERT L., JR.	604 Sixteenth St., McKeesport, Pa.
MATLACK, E. V.	Westminster Apts., St. Louis, Mo.
MAYNARD, T. POOLE	Hart Bldg., Atlanta, Ga.
MILLER, HERBERT F., JR.	P. O. Box 142, Verona, Pa.
MORROW, JOHN T.	C. H. Boynton, 60 Broadway, New York, N. Y.
MORSE, GEORGE H.	Tunnelton, W. Va.
MURRAY, H. T.	Care Amer. Smelt. & Ref. Co., Hayden Plant, Hayden, Ariz.
NEEL, CARR B.	Sebastopol, Cal.
NICHOLS, H. W.	Field Museum of Natural History, Chicago, Ill.
NORRIS, R. V.	Second National Bank Bldg., Wilkes-Barre, Pa.
ORR, CHARLES T.	Pres. and Genl. Mgr., Bertha A. Mining Co., Webb City, Mo.
PATRICK, W. B.	R. F. D. 1, Littleton, Colo.
PELTON, HAROLD A.	1125 Central Ave., Los Angeles, Cal.
PETERS, RICHARD, JR.	Sales Agent, Producers Coke Co., First National Bank Bldg., Uniontown, Pa.
POCOCK, CECIL W.	892 Ave. C, Bayonne, N. J.
POLHEMUS, J. H.	New Jersey Zinc Co., 55 Wall St., New York, N. Y.
RANDOLPH, EDWARD.	580 Market St., Newark, N. J.
REA, J. EDWARD.	W. 25 Seventh Ave., Spokane, Wash.
REDFEARN, ALBERT M.	58 King Ave., Detroit, Mich.
REECE, FRED B.	Care Mrs. Reece, "Highfield," Liscard, Liverpool, England.
REYNOLDS, GEORGE B.	Care British Consul, Maracaibo, Venezuela.
REYNOLDS, L. BOWLEY	Instructed to hold all mail.
RODGERS, J. H.	R. 10, Box 146-C, Los Angeles, Cal.
ROGERS, O. P.	5624 Bramble Ave., Cincinnati, Ohio.
RULE, J. ARTHUR.	P. O. Box 482, Tooele, Utah.
SACKET, CHARLES T.	Livingston, Mont.
SCHUECH, W. A.	Southern Cross, Mont.
SCHUETTENHELM, J. B.	Box 1316, Goldfield, Nev.
SCHWENNESEN, A. T.	1007-13th St., N. W., Washington, D. C.
SETHNA, N. R.	Care Midland Coal, Coke & Iron Co., Ltd., Apedale, Newcastle, Staffordshire, England.
SHERWOOD, F. P.	15 W. 67th St., New York, N. Y.
SHUMWAY, R. W.	1102 Foster Bldg., Denver, Colo.
SMITH, EVERETT W.	Assayer, Weedon Mining Co., Ltd., Weedon, Quebec, Canada.
STALDER, WALTER.	1022 Crocker Bldg., San Francisco, Cal.
STERNFELD, THEODORE.	P. O. Box 957, New York, N. Y.
STICHT, ROBERT	Queenstown, Tasmania.
TACKMANN, HENRY.	Granby, Mo.
THROPP, J. E., JR.	Mercersburg, Franklin Co., Pa.
TIMM, J. M.	36 St. Leonhardstrasse, Basle, Switzerland.
TONG, SING KEW.	"Idelwild," Media, Pa.
UTLEY, H. H.	Care A. G. Utley, Phoenix, Ariz.
WALLOWER, F. C., Engr. of Mines	F. C. & E. Z. Wallower, Joplin, Mo.
WANDER, ERNEST, Chem.	Care Western Cartridge Co., East Alton, Ill.
WESTERVELT, E. W.	1545 St. Paul St., Rochester, N. Y.
WOODS, CLARENCE.	Supt., Louisiana Development Co., Tuolumne, Cal.
WRIGHT, LOUIS A.	Cons. Engr., General Development Co., 61 Broadway, New York, N. Y.
WUENSCH, C. E.	Care Empire Zinc Co., Canon City, Colo.
YATES, A.	46 A. Harrington Gardens, South Kensington, London, S. W., England.
YATSEVITCH, M. G.	60 Kirkland St., Cambridge, Mass.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
CHEVRILLON, LOUIS	Apartado 18. Bis., Mexico City, Mexico.
CRANDALL, RODERIC.	Fazendas Nacionais do Rio Branco, Amazonas, Brasil.

GOEDICKE, CARL.....	Box 535, San Antonio, Texas.
GOODLAND, GILMORE.....	17 Gracechurch St., London, E. C., England.
JONES, EDWARD B.....	Box 502, Salt Lake City, Utah.
MENTZEL, CHARLES.....	New York, N. Y.
NEWBERRY, R. W.....	Box 1477, Bisbee, Ariz.
PETERSON, FRANK.....	622 I. N. Van Nuys Bldg., Los Angeles, Cal.
WHITE, R. T.....	Dzansoul, Russia.
WRIGHT, PERCY E.....	Seattle, Wash.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Dec. 10, 1914, to Jan. 10, 1915:

Date of Election	Name	Date of Decease
1892	*Baer, George F.....	April 24, 1914
1907	*Gordon, Lyman F.....	December 20, 1914
1890	*Hall, Charles M.....	December 27, 1914
1906	*Wood, Stuart.....	March 2, 1914

*Member.

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 EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
 CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
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R. C. GEMMELL, *Chairman*. C. W. WHITLEY, *Vice-Chairman*.
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P. A. MOSMAN,

CHARLES F. RAND,

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GEORGE C. STONE,

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H. C. HOOVER,

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¹ Until Feb., 1915.² Until Feb., 1916.³ Until Feb., 1917.⁴ Until Feb., 1918.

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¹ Until Feb., 1915.² Until Feb., 1916.³ Until Feb., 1917.⁴ Until Feb., 1918.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

German and Other Sources of Potash Supply*

BY CHARLES H. MACDOWELL, CHICAGO, ILL.

(New York Meeting, February, 1915)

Up to 1909 the American public had little knowledge of, or interest in, potash. Some remembered that it had to do with soft soap and sore throat, but further they knew not. In 1909-10, the German-American potash war, and the publicity attending hostilities, brought home to America the knowledge that potash was an important article of commerce; that Germany monopolized its production and distribution, and exercised all the prerogatives attending complete control. The present war, with its transportation embargo, has again directed attention to Germany's strength as a potash producer and her present unfortunate situation as a distributor. As a consequence, the search for potash takes on new life and it again becomes a subject of general interest.

My first knowledge of potash came from a study of the German potash exhibit at the World's Fair in 1893. This investigation led to the recommendation that Armour & Co. engage in the manufacture of commercial fertilizers and the fertilizer branch was organized early in 1894. From then to now "potash" has been a live subject.

Two and eight-tenths per cent. of the earth's crust consists of potassium oxide. It is an important constituent of feldspar, granite, and other igneous rocks. Potash is valuable commercially in a water-soluble form. In igneous rocks it is locked in so tight that it is of little value and no utilization in a commercial way has been made of these insoluble forms. Potash in water-soluble form is found in large deposits in Germany; in smaller deposits in Austria and Spain; and in brines in some of our Western arid basins. The deposits in Germany, Austria, and Spain are concentrations from sea waters containing potash derived from the decomposition of potash-bearing rocks.

It has been found that the amount of potash remaining in decomposed rocks is not the same as that in the original rock. For instance, granite containing originally 5.58 per cent. potash only contained 1.49 per cent. in the decomposed rock. Syenite containing 6 per cent. in the original

* Presented at a meeting of the Chicago Local Section, Dec. 12, 1914.

rock only contained 0.23 per cent. in the decomposed rock. Soil also loses potash.

The probable explanation for this loss of potash is the passing of the rain and soil solutions over constantly renewed exposures of the rock, washing out the soluble portion of the potash, which eventually finds its way into the rivers and finally to the sea. It is estimated that one cubic mile of the river waters of the United States contains 20,358 tons of potassium sulphate. It can thus easily be seen that an enormous amount of the available soil potash is being transported by the rivers and deposited in the sea. The sea brines containing salt, potash, magnesium, and other salts on concentration drop out the less soluble salt, the mother liquor containing the potash accumulating and later desiccating or crystallizing as veins through rock-salt deposits, over which are formed water-proof covers protecting the deposit and holding it for future use. This briefly is the legend.

German Deposits and Mining Methods

Potash salts are found in Germany in the Stassfurt, the Hannover, the South Harz Mountain and the West Alsatian sections. Mines are also operating close to Hamburg and Bremen, and lean deposits, with poor cover, are said to extend into Holland. The deposits near Muelhausen are within 25 miles of the French frontier. In the Stassfurt and Hannover sections, the deposits are often found in salt saddles of great depth. The cover has been disturbed and water dangers are present. Several shafts have been lost by flooding. In the South Harz and Alsatian sections the layers are more nearly horizontal; the cover has been little disturbed and water troubles are less likely. The depth of the potash and salt deposits from the upper to lower stratum is some 5,000 ft.

Prospecting is done by core drilling. A saturated salt solution is used for drilling, which permits the extraction of a full-sized core of the soluble salt. This method was first used in locating the potash beds of the Aschersleben mine, one of the oldest and largest producing mines. Potash beds have been uncovered in prospecting for oil, coal, and gas. Boring companies are formed; acquire mining rights; prospect; and dispose of proved properties to companies who sink shafts and install concentrating plants.

In Prussia the land owner originally had all the mining rights, while in Hannover the mining rights belonged to the first discoverer of the salt. These laws, of course, have been greatly modified in developing the mines in recent years. A royalty of 60 pfennig per 1,000 kg., or say 15c. a metric ton crude, is generally paid the owner of the rights. Operating companies are organized either as *Gewerkschafts*, with 1,000 *Kuxen* or shares of no par value, assessments being made from time to time to complete the work under way; or as *Actien Gesellschafts*, with a fixed capital in shares of a par value of 1,000 marks.

In shaft sinking the Kind-Chaudron system is generally used. Freezing is resorted to if quicksand is encountered. Shafts are sunk by contract. The average cost of shaft sinking is about \$150 per foot. Shafts vary from 300 to 900 m. in depth.

The first potash discoveries were made in Stassfurt. The Stassfurt salt mines are first mentioned in literature in 1227. In earlier years they were used as sources of common salt. Early in the 19th century they were sold to the Prussian government, which started the first bore hole in 1839, striking salt in 1843. The first water that was pumped up from this bore evaporated to about 99 per cent. salt, but it soon became very impure with magnesium and potash salts. The first shaft was commenced in 1852 and struck the salt layer in 1857. It was not until 1860 that Professor Francke first advanced the idea of the various strata in salt mines being of importance for other products than salt, and it was in this year that potash was first prepared.

My first visit to a potash mine was in 1906—the mine Sollstedt, near Nordhausen, shipping crude salts and manufacturing muriate. No sulphate was made there because the salts contain no kieserite. Some notes taken at that time may be of interest:

"Shaft sinking began October, 1902; finished June, 1904; depth, 705 m.; diameter, 8 m.; iron tubing in sections to salt layers, wood below (many mines use brick through salt layers). Mine dry. Hoisting engine Magdeburg make, 1,000 h.p.; cages double decked; shaft head, iron with concrete thrust blocks. Takes five minutes to descend. Considerable jump to cage at lower levels due to cable stretch. Ventilating fan run by horizontal compound engine of 10,000 cu. ft. per minute capacity. Engine made at Aschersleben, novel valve motion. One direct-connected D. C. motor for lighting and electrical power purposes. Power requirements, 1,600 h.p. all told. Burn lignite; mined in neighborhood; little smoke. Separate ventilating tunnel and shaft connecting with main shaft about 75 ft. from bottom. Cover rock largely reddish sandstone. There is a layer of salt clay, anhydrite, and rock salt above the potash deposits, protecting the latter from water danger. Color of potash salt (hardsalt), reddish interspersed with blackish anhydrite and yellow rock salt, these latter layers an inch or so wide. Average potash contents of veins 16.5 per cent. K_2O ; thickness, 10 m. Salt hard and dry. No timbering necessary. Mining under government control. Large chambers laid off with 5-m. supporting pillars. Chambers are cut out to height of 8 ft., width of 20 ft.; potash blasted down and transported in 1-ton cars to shaft. Pillars are permanent, chambers filled in later with mill refuse. Electric and hand drills used. Cars are hauled up grades by electric traction. Mining rights purchased from owners of land on royalty and percentage-of-profit basis. Mine controls right 2.5 miles in three directions. Property abuts Bleichroder mine owned by Prussian government. Deposits figured sufficient to last 800 years.

"Shaft $\frac{1}{4}$ mile from mill, located on railroad; transportation, drag chain on industrial track, chain dropping over bail of car; no clutch necessary; speed, $2\frac{1}{4}$ miles an hour. Salt, when elevated, conveyed to mill building; dumped to screen over crusher, car being turned upside down. Movable screen bars about an inch apart. These bars work from an eccentric shaft; screen out fine particles and move coarse pieces to crusher. The crushed salt goes to the cookers for concentration, or to the mill for shipment as crude in bulk. Cookers are steel tanks on order of pressure tanks with

agitators. The potash is dissolved in a hot solution of magnesium chloride, known as the mother liquor, leaving the salt undissolved. The dissolved hot solution is concentrated and run to large crystallizing tanks, where, on cooling, the potash crystallizes out, the mother liquor being pumped out, heated by exhaust steam and used over again. The waste salt is used for mine filling. Iron crystallizing vats are arranged in clusters of four, surrounded by cement-covered aisles in which are laid industrial-railway tracks. These crystallizing vats cover a considerable acreage. The crystals are shoveled from vats into cars, then to a washing and draining platform, then to storage. Concentrated salts are shipped in bags; mine-run salts in bulk."

This description answers well for other mines, except that in carnallite mines the mother liquors are not used over again and must be discharged into sewers to go to the rivers. On this account, carnallite mines can only manufacture a certain predetermined tonnage of concentrated salts; their final liquors are metered, and operations must be curtailed when the river water attains a certain hardness. Carnallite mines make the bulk of the sulphate of potash manufactured, owing to the presence of kieserite necessary for the production of sulphate. Some mines have chemical manufacturing subsidiaries, where electrolytic and chemical processes are used to manufacture other forms of potash and potash combinations.

The method of mining varies according to the way the strata lie in different mines: Where the layers are anywhere near horizontal, continuous ordinary stoping is resorted to, and they usually drift on the bottom of the seam, blasting down from the top and proceeding in all directions from the shaft. Some mines, however, are operated by drifting at the top surface of the stratum and digging out underneath the level. Where the various layers approach the perpendicular, they sink shafts in the material, using inclined skipways to the main shaft. When the deposits are in lenses, "gobbing" is resorted to, filling in with common salt. Power houses and other plant buildings are solidly and expensively constructed. The workman's comfort and safety are thoroughly looked after. Government supervision is everywhere apparent.

Cost of Mine and Concentrating

Plant.—A mine encountering no unusual difficulties in shaft sinking, and with a shaft depth of say 750 m., with plant site, rail connection, power plant, chemical plant for concentrating, workmen's houses, and all necessary equipment, would cost from \$1,250,000 to \$1,500,000. A second shaft would cost about \$350,000 additional. A fully equipped two-shaft mine, of an annual capacity of say 12,000 to 15,000 tons pure potash, would cost \$1,750,000 to \$2,000,000.

Cost of Production.—This naturally varies with the quantity and kind of salt mined. Owing to the large number of producing mines, I would say that 36 hr. a week would be average working hours, so the output per

mine is small. The cost of mining carnallite averaging 9 per cent. K_2O and delivering it to the mill is approximately 4 marks, or say \$1, a metric ton of 2,204 lb.; hard salts (16 per cent. K_2O) would cost 5 marks, or \$1.25 per ton. I estimate that a good mine, producing 7,500 tons of pure potash per year, can produce muriate of potash (50 per cent. K_2O) at \$19 per metric ton; kainite (12.4 per cent. K_2O) at \$2.40 per metric ton. On a production of 20,000 tons pure potash a year, the same mine could produce muriate at \$14 and kainite at \$1.60 per metric ton. The overhead charges are included in both estimates. All costs are f.o.b. cars. These costs are probably lower than the average mine cost under present conditions. Some hard salts mines, on 24 hr. a day mining, claim to have made muriate as low as \$10 per metric ton. Restricted mining and short working hours now prevent any such costs being attained. Freights to coast during river navigation average about \$1.50 per metric ton, and to American ports \$2.50 to \$3.00; or say \$4 to \$4.50 per metric ton from mine to American Atlantic ports.

Commercial Conditions

Aside from remote water danger, potash mining is unusually safe mining.

Some years ago, because of overproduction, and with the hope of discouraging new mines on account of expense, a two-shaft law, intended for coal mining and in the interest of safety to miners, was made to apply to potash mines, a reasonable time being given old mines to sink second shafts. The result was unsatisfactory, as old companies would split their holdings, form new companies, sink shafts, and demand new quotas.

For many years, overproduction was the rule and profits the exception. Prussia and other States owned mines. Finally a selling syndicate was formed, all of the then producing mines, including those government owned, becoming parties. Participations in the total sales were agreed on, percentages being figured in thousandths per cent. The syndicate had a life of five years and was headed by a government official. Monetary penalties for violations were exacted. A new syndicate had to be arranged for six months before the expiration of the old syndicate. New mines opening during the syndicate term were taken in and new percentages fixed. Occasionally new mines outside the syndicate would sell below syndicate prices, generally to Americans, but would later enter the syndicate.

On June 30, 1904, one mine refused to enter the new syndicate and for several hours sold freely of bulk salts for five years, finally entering the syndicate subject to keeping such sales as had been made. These sales disturbed the syndicate and steps were taken to make such selling and buying unpopular and unprofitable. Prices on bulk salts were lowered. These reductions automatically lowered the price of the outside mine

contracts, as prices were guaranteed. Five-year exclusive contracts were put in effect. Those having bought outside bulk salts were not to be given lowest discounts on bagged goods. They were to be punished financially for having made outside purchases. Secret discounts were given certain favored buyers. Some of the offending Americans "stood pat," and refused to be punished. In due time a new mine opened up and sold these Americans at lower-than-syndicate prices and for a long term of years, thus defeating the syndicate plans.

In May and June of 1909 I visited Germany and made a thorough inspection of several mines, with the idea of purchase. At that time the various syndicate members were negotiating for the renewal of their syndicate, which expired June 30 of that year. Rumors of serious dissensions were prevalent and the prophecy was made that the syndicate would not be renewed. There were too many mines and too little business. The prophecy proved true, as, at midnight of June 30, members could not agree and free trading was in order. There were several American buyers in Berlin, and, for two hours, a considerable business was effected by seven mines at a material reduction from old syndicate prices and for a term of years. The Prussian government then exerted sufficient influence to stop further trading, for the time. It immediately became known that a large business had been done and mines not participating began to exert pressure to force sellers to cancel their contracts. Thus began the potash war of 1909-10. A Cabinet member discussed the situation in the Reichstag. The press took up the refrain and the stage was set for an interesting commercial comedy. The first suggestion was to levy an export tax on these particular American contracts, but that was decided to be too openly discriminating. One of the contracts contained a clause reading, "Any import or export tax or other governmental charge should be paid by the buyer," meaning any uniform tax, applicable to all business. Here was a legal peg: "Other governmental charge." A law was introduced, and finally passed, declaring potash to be a monopoly; that every producer was entitled to a certain percentage of the total business (the percentage to be fixed by governmental agencies), and that any producer selling more than his fixed percentage should pay a super-contingent tax amounting to more than the purchase price of the contracts. This, of course, only applied to mines selling to America, who had sold large tonnages, and could only apply to American purchasers. Our State Department protested vigorously, and a tariff war was threatened. The situation was strained. For some months these taxes were paid by the buyer under protest. Finally the Americans accepted the situation and a peace was concluded. The amount involved ran into many millions. As a part consideration for canceling the contracts, a percentage of the tax paid under protest was refunded American buyers by the German government. Friendly relations now exist.

During the hostilities, I made the suggestion to the Washington authorities that our government should make a search for potash. Appropriations were made, and the investigation is still under way.

On July 1, 1909, there were 52 producing mines in Germany. Our latest reports are that 160 mines are now producing and from 30 to 40 are under construction. This condition has naturally followed the monopoly legislation and fully \$150,000,000 has unnecessarily been invested in the German potash industry. Thirty of the best mines could take care of the world's needs.

United States' Consumption of Potash

Statistics of the year 1912 show 11,164,000 tons of crude potash mined in Germany. This is equivalent to about 1,100,000 tons of pure potash. Of this, the United States takes from 245,000 to 250,000 tons. The fertilizer industry uses from 235,000 to 240,000 tons and the chemical industry, 12,000 to 15,000 tons. The average monthly arrivals have been about 20,000 tons. From Aug. 1 to Dec. 1, 1914, some 10,000 tons of pure potash have come in, as compared to a normal receipt of 80,000 tons. Burlap bags are now becoming scarce, and concentrated salts will have to be shipped in casks, or taken in bulk, if shipped. The outlook is not favorable for heavy winter shipments. The fertilizer manufacturer probably has enough on hand to carry him through his spring season on a restricted-percentage basis. The chemical manufacturer is in about the same situation. Should the war continue through the summer, the present rate of shipment, if continued, would take care of only a small part of our necessities.

Other Sources

Austria.—Small, partly developed deposits exist in Galicia, Austria, near Kalusz. Not to exceed 1,000 tons a year of pure potash has been produced—Austria importing the bulk of her requirements from Germany. It is probable that German influence has prevented the development of these deposits. It would not be good sense for any great investment to be made in Austria to produce potash when one considers the close relations existing between Austria and Germany and the large investment Germany has made in the potash industry.

Spain.—Some three years ago, we received from Europe a mining engineer's report on a potash discovery made in connection with some salt developments in Spain, near Sauria. Later the discovery became public. Analysis of the salts found showed a good proportion of potash, fairly amenable to refining. Mining reports and maps submitted to us led me to believe that, owing to shallow depths and poor cover, shaft mining in the deposit reported on would be dangerous. Brining might, however, be resorted to. Since that time, further discoveries have

been made; concessions have been granted to companies to prospect; and Spain may become a potash-producing nation. That country should be a good distributing center, and, if the government does not, by export taxes or other restrictive measures, kill the goose that lays the golden egg, Spanish potash may come on the market. This cannot be for several years, however, and does not help in the present predicament. The potash seems to be there.

Peru.—I have examined reports showing fair-sized and workable deposits of nitrate of potash in Peru. One of these deposits has been worked in a small way. It is close to transportation and should be further investigated.

American Possibilities

Some potash can be made as a by-product from other manufacturing industries. Practically no progress has been made in this direction in this country.

The sugar-beet tonnage in the United States totals 5,147,000 tons. Sugar-beet wastes from this tonnage could produce 30,900 tons of potassium carbonate, equal to 15,440 tons of pure potash.

Raw wool contains up to 20 per cent. potash salts. The wool clip of the United States weighs 298,000,000 lb. In wool scouring 7,450 tons of potassium carbonate or 4,470 tons of pure potash is washed out and lost. This could be largely recovered. Much raw wool from Australia and other countries is imported, and the potash from the handling of this wool could also be saved.

Potash from Cement Burning.—Many cements before clinkering contain small percentages of potash, say 0.1 to 0.25 per cent. In burning this is volatilized. While the percentage is small, the tonnage is so large it figures quite a production on paper.

Many experiments have been made to recover this volatilized potash. It has been worked out to a point where the physical part of the operation is a success. It consists of condensing the volatilized potash in chambers and dust collectors similar to the bag house used in smelters. Up to the present time, however, no cement manufacturer has produced potash as a by-product. Possibly the investment expense attached to its recovery figures against it. I have no data on this.

The industry is a possible source of a considerable tonnage of potash and in case of necessity can be utilized without any extensive changes in present methods of manufacture. Possibly cheaper recovery methods may be found.

At one time considerable carbonate of potash was derived from the ash from burning cotton hulls. These are now used for fodder, so no potash could be obtained from that source.

A few years ago tobacco stems were distilled destructively for nico-

tine and other products. The ash from this material contained high percentages of carbonate of potash. This method of handling has now been discontinued, but the potash in the ground tobacco stems is saved and used for fertilizing purposes.

This about exhausts the by-product possibilities as far as I know.

Hard-wood ashes and other ashes are gathered and used in fertilizing for their potash contents. City garbage contains appreciable percentages of potash. This garbage in many cities is recovered and used for fertilizing purposes.

Government Work

Following the 1910 appropriations, the Geological Survey and the Department of Agriculture immediately began their work. One of the first suggestions made was that the giant kelps of the Pacific Ocean might be a source of supply. As far back as 1904 the manager of our Pacific Coast fertilizer business called our attention to these kelps and their large potash contents when figured on a dry basis. He proposed to burn them, recover the nitrogen and iodine, and use the ash. He went as far as to gather a few tons of kelp and burn it. Later we did some research work in Chicago, not only on the Pacific but also on Atlantic Coast kelps, with a view to complete utilization, with rather satisfactory results. We decided, however, at the time that the cost of gathering the wet kelp and handling it made it a rather difficult thing to figure out profitably; that we were dry-land people and a fleet of boats would be needed to gather any tonnage; and that there was little chance of protecting any process we might develop. Further, it was so easy to get the German product, that the matter was dropped. The total tonnage is no doubt large, but the kelp has a small per-ton value when wet and its gathering and preparing present a problem difficult to solve with financial success. A McCormick may come along with a kelp reaper, presser, and binder that may give the solution. The government has done splendid work in charting the beds and in obtaining other data. These beds are more or less continuous from Vancouver to San Diego, with several important concentrated areas, one of which is off the port of Los Angeles and another off the coast of Seattle.

Authorities differ as to the relative importance of these two beds, but apparently those along the northern coast are more important than those to the south.

Little real work has been done in the harvesting and utilization of this kelp. Several stock-selling companies have been formed, and a company operating at Long Beach, Cal., is producing about 500 lb. of combined salts per day. The general plan has been to cut the kelp and allow it to wash to shore, and although some success was obtained with a harvesting boat, no attempt has been made toward extensive operations.

The question has been gone into quite thoroughly from time to time, and unless definite outlets can be found for by-products such as iodine, tar, and so-called gums, it is doubtful if the utilization of kelp can be made commercial.

A second source of potash on which the government has reported is Searles Lake, Cal. At this point there is a large dry lake of several square miles in extent, upon which a company is now erecting an experimental plant for the production of the various salts found in the brine underlying the surface.

This brine is pumped from wells sunk through the solid crystals and contains about 35 per cent. solids, 5.35 per cent. in the form of potassium chloride. The plant which is now under construction is expected to be in operation by Jan. 1, 1915, and this plant will produce daily: 5 tons of potassium chloride; 5 tons of carbonate of soda; $2\frac{1}{2}$ tons of borax; 15 tons of common salt; 10 tons of sulphate of soda.

The company has in mind an ultimate plant that will be 100 times this capacity, to be constructed in units similar to the plant now under construction, so that it will be some time before any considerable quantity of potash will be produced. The supply of brine is apparently large, and under the proposed plan should last for a good many years. The entire success of the undertaking would seem to lie in the outlet for the various salts produced, besides potash. I have followed the development of this project from its inception and it looks to me as if it should work out and become an important source of supply. The development is in strong hands financially.

Alunite.—A deposit of alunite located near Marysville, Utah, is described in *U. S. Geological Survey Bulletin*, No. 511. Alunite is a double sulphate of potash and alumina with the following theoretical composition: Al_2O_3 , 37 per cent.; SO_3 , 38.6; K_2O , 11.4; and H_2O , 13 per cent. The Custer-Marysville deposits are quite pure for large deposits and rather closely approximate the theoretical. A calcining process followed by leaching releases about 85 per cent. of the potash as sulphate, as reported by the government. Large quantities of hot water are required for leaching. Owing to transportation difficulties and to the necessity of finding a market for a big tonnage of impure alumina, the property has not yet been developed. Large capital expenditure would be required, which would hardly be justified from the potash standpoint alone.

Potash from Feldspar.—The first recorded work on the recovery of potash from feldspar was done by Tilghman about 1845; the first United States patent was obtained by Bickel in 1856. Various investigators have been working in this field practically ever since, receiving stimulus from time to time as the potash question became acute.

In recent years, research on this subject has been along four general lines: First, by the use of natural agencies, such as heat, bacteria, carbon

dioxide, etc.; second, wet chemical methods, such as solution in alkalis or acids; third, dry reactions in which the potash is volatilized; fourth, dry heat reactions in which the potash is converted into a water-soluble form. All of these methods have been more or less successfully operated in the laboratory, but up to date their cost has been prohibitive to commercial development.

The fact remains that there is practically an unlimited source of potash in this country in the igneous rocks, particularly those of the feldspathic type, and no doubt eventually some of these processes will be made commercial, probably through a proper usage of the by-products produced in the recovery of the potash.

The use of feldspar in connection with the manufacture of white cements seems to me to be the most encouraging. An Austrian process looks rather good as far as it has gone. The cost of mining feldspar is considerable. A cheaply mined deposit and the necessity of having this deposit in a locality suitable for the manufacture and distribution of cement would be the controlling features in such an industry.

Leucite.—The leucite hills of Wyoming contain large quantities of potash in an insoluble form. As far as I know, no serious work has been done on this material. It is claimed that by a simple process the potash can be made available for plant food, but in the form offered it is not soluble in water and could not be used in the fertilizer industry, as the various State laws require water-soluble potash. Dana gives an analysis of leucite from the Albani Mountains, containing 21.48 per cent. of K_2O . The leucite hill deposits, according to an analysis from the Armour laboratory, contain 11.34 per cent. K_2O . Other analyses of different samples show even higher potash contents. The quantity is practically inexhaustible.

Other Brines.—The brines of several alkali lakes and ponds in western Nebraska contain appreciable percentages of K_2O . A government analysis of one of these showed 3 per cent. K_2O in the water. An analysis of a sample of the solids from one of these lakes made by the Armour laboratory showed 15 per cent. K_2CO_3 and 33 per cent. K_2SO_4 . The Geological Survey reports in substance:

A number of alkali ponds and lakes existing in Cherry, Sheridan, Morrill, Garden and Boxbutte Counties, Nebraska, have been examined and in some of these the percentage of potash is as high as 30 per cent. of the soluble material. Although the total potash in solution is large it is disseminated in the muds so that profitable extraction will prove difficult. Some of the deposits are near transportation.

Within the last two years, well drillers in Texas have reported the presence of appreciable quantities of potassium in brines at depths around 800 ft. Dr. Carl Scholz, of the Rock Island Railroad, reports that from these drillings he is led to the belief that east of the western boundary

line of Oklahoma, in the vicinity of Magum, a potash deposit may lie within easy reach.

Development work, principally deep-well drilling, is also being done in various sections of the country, especially in the West and Southwest. In most cases the object is to find dried-up lake beds containing soluble potash salts or brines from these sources as well as those draining potash-bearing rock territory. The commercial utilization of such workings will depend, of course, upon the extent of the potash-bearing areas and their concentration.

Summary

In by-product lines, the sugar manufacturer and the wool scourer might spend some money on solving their problem. The cement manufacturer could well afford to investigate potash manufacture from potash-bearing rocks in connection with cement manufacture.

If potash is to be produced in this country in quantity, some one must spend some money. There has been a noticeable reluctance on the part of American capitalists to finance serious research work, or prospecting by deep core drilling. To begin with, our mining laws and mineral withdrawal possibilities are discouraging. A German potash mine controls several thousand acres of mineral rights. The necessary capital investment would not be justified if this control was not legally obtainable. Potash mines with the necessary concentrating plants are expensive and a large acreage control is necessary. Deep core drilling or other development work on government lands by private individuals would not be justified during the present stage of development of the art of conservation as practiced by our legislators. A positive control of acreage must go with such preliminary work.

There is a large market for potash and the demand is growing. Possibly the lever of necessity may develop sufficient power to overcome the prevailing inertia. Up to now Searles Lake is the only nearby producer on the map.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Investigation of Sources of Potash in Texas

BY WILLIAM B. PHILLIPS,* AUSTIN, TEXAS

(New York Meeting, February, 1915)

THE possible sources of potash salts in the United States have been considered from many points of view during the last several years, but it is only within the last two or three months that the situation has become acute. We import from Germany, the only source of large supply, more than \$10,000,000 worth of potash salts annually, and this business has been badly crippled by the European war.

If there was ever a time in our industrial existence when the necessity of providing for our own needs was more acute than it is now, we are not aware of it. One has only to study the price lists of chemicals sent out by the large supply houses to see the change wrought within a very short time. Whether or no these prices are to maintain for a considerable time, no one can say; but it appears to be probable that we shall not see prices materially lower for a period of uncomfortable length, to say the least.

It is not my purpose at this time to discuss the various sources of supply of potash salts that have been suggested or recommended. Some of them might afford a pleasing relief but all of them combined would not affect the situation seriously. If we are to have our own sources of potash salts, they will have to compare, in some efficient manner, with the sources upon which we have drawn during the last 20 or 30 years; *i.e.*, they must exist in minable quantities.

It seems to me to be quite idle to speculate upon our domestic sources of potash if those sources are not of a character similar to those upon which we have so long depended. A few thousand or, perhaps, a few hundred thousand dollars' worth of chemically derived potash may be produced under the stress of present conditions, but when we consider that each year demands more than \$10,000,000 worth, we must look to underground sources; we must look to mines similar to the famous Stassfurt and Halle deposits. If we cannot mine potash salts, we shall have to do without them for a while.

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Is there any general locality within the United States where deep boring for potash salts can be recommended? Is there a working chance anywhere?

This is an important question and one to which a satisfactory reply can hardly be made now.

I venture to mention some circumstances that encourage me to believe that certain parts of Texas are distinctly within the possibilities. I must not be understood as saying that I believe potash salts in working quantities do exist in Texas, but I do believe that the chances are well on the affirmative side. These circumstances are as follows:

In the deep well at Spur, Dickens County, 200 miles northwest of Fort Worth, from a depth of 2,000 to 2,200 ft. was obtained a very salty water which carried 324 grains of potassium chloride per U. S. gallon. The composition of this water was as follows (analysis by S. H. Worrell, Chemist to the Bureau of Economic Geology, University of Texas):

	Grains per U. S. Gallon
Calcium sulphate.....	1,406.19
Calcium chloride.....	679.02
Magnesium chloride.....	219.20
Sodium chloride.....	3,410.55
Potassium chloride.....	324.14
Total.....	6,039.10

In a recent publication,¹ J. A. Udden, who carefully examined the borings from the 4,489-ft. well, and collected samples of water from it, reports on the potash content (as chloride) of 15 samples of water taken from depths ranging from 800 to 3,000 ft. below the surface. The minimum amount of potash in grains per U. S. gallon was 21.4, in drip water from 1,550 ft.; and the maximum amount was 324.1 grains, at a depth of 2,200 ft., the average being 77.7 grains. The table of analyses submitted is given on the opposite page.

In commenting on these results, Dr. Udden says:

"The differences in the analyses made here indicate the presence of a potash-bearing stratum somewhere near 2,200 ft. below the surface in this well. The sample having 324.1 grains per U. S. gallon was taken when work on the well ceased for a while, in April, 1912. All the other samples were taken two months later, before the work of drilling was resumed. The water had been standing undisturbed for two months, and the potash-bearing ingredient in the entire seepage had evidently been more evenly distributed through all the water in the hole."

¹ The Deep Boring at Spur, *Bulletin of the Bureau of Economic Geology, University of Texas*, No. 363, p. 83. See also an article by the same author, in the *American Fertilizer*, December, 1912.

Potash Content (as Chloride) in 15 Samples of Water taken at different depths in S. M. Swenson & Son's Deep Boring at Spur, Tex.

A	B	C
800	800	72.2
1,360	300	59.1
1,390	1,390	61.9
1,500 (drip)	1,550	21.4
1,600	1,600	42.9
1,800	1,390	94.4
1,830	1,830	47.2
2,000	2,000	41.0
2,200	2,200	324.1
2,200	1,500	113.3
2,300	2,300	60.2
2,400	2,400	69.6
2,600	1,500	86.2
2,690	2,400	29.1
3,000	1,500	43.6

A. Depth at which sample was taken, in feet below surface.

B. Depth to surface of water, when sample was taken, in feet below surface.

C. Grains of potassium chloride per U. S. gallon.

Tests for potash in the cuttings were made on 28 samples, representing material taken from depths ranging from 732 to 2,392 ft. From 2,042 to 2,047 ft. there was a trace of potash. The material was a blue shale, effervescing slowly in acid. From 2,068 to 2,110 ft. there was a pronounced trace of potash. The material here was a deep brown, highly ferruginous and slightly micaceous sandy shale, and dark gray, soft anhydrite-bearing dolomite. With these two exceptions, none of the cuttings gave a trace of potash. In commenting on the tests made on the cuttings, Dr. Udden says:

"It can hardly be a mere coincidence that traces of potash were found in the solid rock only near the level where the potash content was highest in the well water. The very unusual amount of 324 grains per gallon also strongly suggests the existence of more than a mere slight impregnation in some stratum. Even after the potash had dissolved from the exposed walls of this stratum and had been diluted through a column of water 3,000 ft. high, it was present in an amount averaging 60 grains per gallon for all the water in the well."

The well at Spur is the deepest well ever bored in Texas. It reached a total depth of 4,489 ft. The elevation at Spur is 2,274 ft. above sea level. The upper 1,250 ft. represent the Red Beds of the Permian, a part of the Double Mountain formation. The next 2,850 ft., of dolomite, anhydrite, sandstone, and shale, are taken to be the equivalent of the Wichita, Albany, Clear Fork, and (in part) the Double Mountain, and possibly are correlated with the Delaware formation west of the Pecos River. The

lower 389 ft., limestone and shale, are thought to correspond to the Cisco (Carboniferous) formation of central Texas.²

Three beds of pure salt were found in this boring, five-sixths of it being in the lower half of the Red Beds. One bed, 10 ft. thick, was at 570 to 580 ft.; another, 5 ft. thick, was from 633 to 638 ft., while another, 9 ft. thick, was from 732 to 741 ft. Salt was also noticed at about 1,600 ft. and again at 2,244 to 2,264 ft.

Under the heading "Prospecting for Potash," p. 87 of the *Bulletin* referred to, Dr. Udden remarks:

"Considering the great value of a workable deposit of potash, it seems worth while to call attention to another circumstance in connection with these observations. In either direction, north or south, from Spur, the formations lie practically horizontal for at least a hundred miles, and the potash-bearing formation, whether it be such or not in other places, must be at about the same depth as here, in these directions. It seems to the writer that the general conditions indicated in this boring, the existence of great salt beds and beds of anhydrite, together with the proven potash-bearing stratum, warrant an examination for potash in water from the same horizon in any boring made in this territory. East or west from Spur the depth to this horizon will be greater westward and less eastward. The elevation of the railroad depot at Spur is 2,274 ft. above sea level. This is 666 ft. higher than the elevation at Cisco, about 120 miles to the east-southeast. A line connecting these two points may be taken to follow the direction of the general dip of the formations to the west. The bottom of the well may be taken to represent the beds outcropping at Cisco. On this assumption, the general dip between Cisco and Spur, a distance of 120 miles, will be equal to the depth of the Spur well less the difference in elevation of the two places. This gives us a dip to the west of nearly 32 ft. per mile. Our inability to fix the precise level in the Cisco formation reached in the boring may make this figure either a little too high or too low, but it cannot be far from right. Taking into consideration the general east slope of the land surface, which averages 6 ft. per mile, any stratum should come nearer the surface at the rate of 38 ft. per mile eastward from Spur.

"Assuming now that this general dip is constant between the two points and that the formations are continuous, the horizon which yielded potash in the Spur well should outcrop in a belt where the land surface intersects the dipping plane lying 2,200 ft. below the surface at Spur. This belt would extend through Haskell and Jones counties. . . . Along the line of the Kansas City, Mexico and Orient Railroad, in these counties, the potash-bearing horizon may be looked for at depths of from 100 to 400 ft. It is not to be expected that potash should be found in any outcropping rock in this belt, owing to surface leaching, but well waters might show its former existence."

As one goes west from Spur and crosses Blanco Canyon into Crosby and Lubbock counties, there are no deep wells from which records are available, nor are any to be had from the counties still farther west, such as Hockley and Cochran. The same is true of the counties to the south-west, such as Lynn, Terry, Yoakum, Gaines, Dawson, Andrews, Martin, etc., lying along or adjacent to the New Mexico border.

Charles L. Baker is now engaged in making a detailed examination of the water resources of Hale County, for the Bureau of Economic Geology

² Udden, *ut supra*.

and Technology, and his report will include the analyses of a large number of well waters from that area, northwest of Dickens County. In the Texas counties immediately southeast of New Mexico there are many old salt basins, half-dried up lakes, etc., which afford not only the usual alkaline compounds, such as salts of lime, magnesia, and soda, but considerable deposits of almost pure sulphate of soda as well. The accompanying photograph (Fig. 1) shows the salt flats at Toyah Lake, Reeves County. It is in such areas that the search for potash salts should be conducted. But boring operations there would involve a large outlay of cash, for it does not appear probable that the cost would be less than \$10 a foot.



FIG. 1.—SALT FLATS AT TOYAH LAKE, REEVES COUNTY, TEXAS.

The cost of the 4,489-ft. well at Spur was \$50,000, or about \$11 a foot, although the salvage reduced the cost to \$45,000. Boring operations in the counties less favorably situated with respect to transportation, fuel, water, and supplies are not likely to be conducted for a less cost than the Spur well and in many localities the cost would be greater. No such operations should be undertaken without a thorough geological examination, and when the cost of this is added to that of the boring and of the examination of the cuttings, water, etc., a most important consideration, the undertaking would have to be backed by ample capital. The State of Texas still owns a good deal of land in those counties, as the following

statement, taken from the report of the Commissioner of the General Land Office, Austin, for the two years ending Sept. 1, 1914, will show:

County	Acreage Held by the Public School Fund
Andrews.....	4,892
Cochran.....	640
Crane.....	11,382
Ector.....	630
Gaines.....	7,167
Loving.....	5,420
Terry.....	1,760
Winkler.....	4,925
Yoakum.....	9,736
	46,552

In addition, the University of Texas owns more than 450,000 acres in this part of the State.

The price of this land varies from \$2 to \$5 an acre for the surface rights. Such lands are sold with reservation of the mineral rights. The mining law passed by the Legislature in 1913 declares it to be the policy of the State not to dispose of the mineral rights in these lands, but to lease them on a royalty basis of 5 per cent. of the gross value of the minerals taken from them. The Public School lands are administered by the Commissioner of the General Land Office, but no provision is made for the examination of such lands with reference to their mineral character. No funds arising from the sale or lease of these lands are, or, under the present law, can be, applied to investigations of a geological or technical nature. Such things are left to private or corporate enterprise. Failing this, there is no source of information available to the public.

Dr. Udden has suggested a search for potash in wells bored to the south-east of Spur, along lines bearing northeast and southwest between the Brazos and Colorado Rivers. In addition to this there should be as detailed an examination as possible in the counties already mentioned, especially in those where extensive deposits of alkali salts are known to exist in old lakes, basins, etc. The more favorable localities for deep boring should be ascertained and mapped, so as to reduce to a minimum the risks attending such operations. If such soluble compounds as common salt, carbonate of soda, sulphate of soda, sulphate of magnesia, etc., are known to exist there, both on the surface and at shallow depths, there is no reason why potash salts, no more soluble than these, should not be found also; i.e., no reason in so far as concerns solubility in water. The average annual rainfall is about 15 in., but so unevenly distributed that for months together there is no precipitation at all. It is during such lengthy droughts that many of the old lakes, basins, etc., appear to be covered with snow, so rapid is the accumulation of salt, etc., due, in part,

to capillarity. Under the winds that blow almost constantly and under a sun that becomes oppressive, at times, evaporation proceeds with extraordinary rapidity. A shallow lake to-day becomes a bed of glistening salt to-morrow.

I have thus far dealt with facts and suggestions which have been ascertained and ventured upon with reference to that part of Texas which lies southeast of and adjacent to New Mexico. There is still another area, bordering upon old Mexico, in the counties of Brewster and Presidio, which has afforded some interesting data with regard to nitrate of soda and nitrate of potash. I have personally examined all of the localities from which these minerals have been reported, and while I cannot say that I have seen anything of commercial value, still, the occurrences are of unusual interest.

In June, 1914, I received from a friend in Alpine, Brewster County, a sample of mineral which was identified as a mixture of nitrate of soda and nitrate of potash. I sent a part of the mineral to Ledoux & Co., New York, and received from them, under date of July 6, 1914, the following analysis:

	Per Cent.
"Moisture.....	0.77
"In the undried sample:	
Potash (K_2O) soluble in water.....	30.46
Soda (Na_2O) soluble in water.....	4.16
Nitrogen (as nitrate).....	7.63
Equivalent to nitric anhydride (N_2O_5).....	29.42

"The sample also contains small amounts of water-soluble lime, chlorine and sulphate radicle; it is, therefore, impossible to state the exact contents of potassium nitrate and sodium nitrate without making a more complete analysis than the character of the sample warrants. But the data given above are sufficient, assuming sodium to be combined as nitrate, to say that sodium nitrate amounts to 11.4 per cent. and potassium nitrate to 41.50 per cent., leaving a remainder of about 11 per cent. of potash (K_2O) combined as other salts. On the other hand, if all of the nitric anhydride is assumed to be combined with potash, the amount of potassium nitrate would be 55.10 per cent., requiring 25.65 per cent. of potash and leaving a remainder of 4.8 per cent. of potash, together with all of the soda, combined as salts other than nitrate."

Upon receipt of this analysis and after securing further information, I visited the locality in August, and obtained many samples. An average of the better grade of material was sent to Ledoux & Co., and under date of Aug. 28, 1914, they reported as follows:

"The sample of earth containing nitrates of potash and soda marked 'Tinja on Sec. 120,' referred to in your favor of the 15th instant, has been examined as follows:

	Per Cent.
Water-soluble salts (by solution and evaporation).....	29.24
Insoluble sandy matter (by difference).....	70.76

On the basis of the original sample, the water-soluble salts contain:

	Per Cent.
Silica.....	0.10
Calcium.....	0.46
Sodium.....	1.00
Potassium.....	9.71
Chlorine.....	1.44
Sulphate radicle.....	1.07
Nitrate radicle.....	15.52
	29.30

"These elements and radicles are probably combined as follows:

	Per Cent.
Calcium sulphate.....	1.52
Sodium chloride.....	2.37
Sodium nitrate.....	0.28
Potassium nitrate.....	24.96
	29.13

"The aqueous solution of the salts contains a trifling amount of organic matter, but it is free from iron, aluminum, or magnesium salts. The composition of the dry salts would, therefore, be practically:

	Per Cent.
Calcium sulphate.....	5.2
Sodium chloride.....	8.1
Sodium nitrate.....	1.0
Potassium nitrate.....	85.7
	100.0

The question of commercial quantities arises at once. I regret to say that from a close examination of the locality and the results of chemical tests of many samples of the rock in place, I am unable to express a favorable opinion.

The potassium nitrate was found as incrustations on, and thin seams in, a porous sandstone of Cretaceous age. The exact locality is on Sec. 120, Block G-4, Brewster County, Texas.³ This section is east of Maverick Mountain about 5 miles, and is between this mountain and the more immediate western foot hills of the Chisos Mountains. The nearest railroad point is Marathon, or Alpine, stations of the Southern Pacific Railway, in Brewster County, from 230 to 260 miles southeast of El Paso. The Terlingua quicksilver district is from 6 to 12 miles west.

The sandstone involved dips to the north and east from 7° to 10°, but is exposed only at points where the surface drift, etc., has been worn away by water. Around Chisos Pen, about 2½ miles to the northeast, this

³ Thin seams of nitrate of potash occur also in a red porphyry in this vicinity.

sandstone carries beds of a sub-bituminous coal which has been used under steam boilers at some of the quicksilver mines.

At the particular place where the nitrate was first found, by a Mexican named Miguel de la O., on Sec. 120, Block G-4, the flood waters of Alamo Creek have cut down into the sandstone and made a fine exposure of 25 to 30 ft. I say the flood waters, for this creek is ordinarily entirely devoid of water except where it has been possible for it to excavate a *tinaja* (water-hole) in the rock. Such a *tinaja*, when well sheltered, may hold water for several months, or from one rainy season to the next. There is a large *tinaja* of this kind at this place and fairly good drinking water may be obtained through the greater part of the year. That this circumstance has an important bearing on the accumulation of nitrate of potash is, I think, evident from the local conditions.

The sandstone beds are of unequal hardness and some of the upper ledges now project over the lower ledges by several feet and afford sufficient shelter to be used by Indians. Under one of the more prominent of these overhanging ledges there is now a considerable pile of ashes, half-consumed pieces of charcoal, remains of crude pottery, arrow-heads, etc. This débris carries a good deal of nitrate, but only in the upper part, upon which the nitrate from the roof has fallen. The roof of this half-cave is incrustated with nitrate, some of it hanging loosely to the sandstone and falling at a slight jar. Running through the roof are seams of almost pure nitrate of potash varying in width from $\frac{1}{8}$ to 1 in., and these appear also at the back of the cave. But behind the incrustations and alongside of the seams there is a mere trace of potash in the sandstone itself. Two inches behind a heavy incrustation of nitrate the sandstone barely reacts for potash, at 6 in. there is no reaction at all.

After removing the incrustation of nitrate with a chisel the sandstone shows no impregnation with nitrate. The seams of nitrate do not enter the rock, but are confined to the surface or to a plane but slightly below the surface.

I think that the nitrate of potash here has not been derived, by capillary action, from deeper-lying deposits. From what we know of the sources of this material elsewhere, we shall have to look to animal excreta, especially human excreta, for its origin here.

It is, I think, beyond question that this *tinaja* was a camping ground for a great many years. The water is well sheltered and abundant for the greater part of the year. The adjacent country is even to this day a favorite range for deer, quail, etc., and the dry bed of the creek affords a comparatively easy and safe route to the Rio Grande, for hunting or war parties.

I made a diligent search for other like places in the vicinity, but found none. Many samples of sandstone, shale, etc., were examined for potash, but not one revealed its presence. It is only at this particular *tinaja* that

any potash has been discovered, and the conditions here seem to me to preclude commercial possibilities, however interesting from a scientific standpoint. Aside from such circumstances as have been mentioned, there is one that militates most effectively against the probability of the existence of workable deposits of potash anywhere near the surface; and that is, the mean annual rainfall in the district. While this cannot be stated accurately, it is not likely to be less than 15 in. This means that upon every square mile there falls every year not less than 285,000,000 gal. of water. During a residence of nearly 18 months in the adjacent quicksilver district, I observed the most awful storms, the eroding, dissolving, and transporting effects of which were most remarkable. The erosion that has taken place at the tinaja and that is now in progress is abundant evidence of the destructive nature of these torrential storms. It is possible that some isolated deposit of nitrates might have been buried under one of the great lava flows which characterize that region, but there is no evidence of such a thing. To base active prospecting upon such a remote possibility is not at all advisable.

The accompanying photograph (Fig. 2) shows the curious effects of erosion at the tinaja and the shut-in character of the tinaja itself. The long well-like excavations in the sandstone are the halves of what were once pot-holes. In places these are 10 to 15 ft. in length and from 4 to 12 in. in diameter. I counted more than 50 of such pot-holes, some of them drilled entirely through an overhanging ledge of sandstone, some of them now in process of drilling, and some drilled for 10 or 15 ft. and then cut through vertically by the water. On the Linville River, western North Carolina, are many beautiful illustrations of pot-holes worn in a sandstone. In a country where the mean annual rainfall is probably not less than 15 in., as in this part of Brewster County, and where the porous nature of the sandstones allows the rapid absorption of water, it would not appear that there has been much, if any, opportunity for the accumulation of workable deposits of such soluble substances as nitrate of soda and nitrate of potash. Some time ago I was called to Presidio County, Texas, to examine a so-called discovery of nitrate of soda. The samples that had been sent in were of an excellent character, running as high as 71 per cent. of nitrate. The locality is about $2\frac{1}{2}$ miles east, a little north, from Candelaria, a little settlement on the Rio Grande, 45 miles south of Valentine, a station on the Southern Pacific Railway.

The nitrate was found as thin veins in a dense, hard trachyte which had been eroded near the top of a small canyon so that cave-like excavations had been formed. A similar occurrence was noted several miles to the west of Candelaria and at this place some old workings were plainly visible. Many samples of the nitrate and of the inclosing rock were examined, but in no case did there seem to be any reason for further prospecting. The thin veins of nitrate are highly localized, and, while rich

specimens were obtained, the commercial possibilities are of no moment.⁴ The accompanying photograph (Fig. 3) shows the situation to great advantage.

Several years ago there were reports of the discovery of nitrate of



FIG. 2.—EROSION OF SANDSTONE AT TINAJA ON SEC. 120, BLK. G-4, BREWSTER COUNTY, TEXAS, WHERE NITRATE OF POTASH IS FOUND.

potash in El Paso County, but investigations showed that while the mineral did exist in almost pure condition, the amount was extremely small.

⁴ See articles by the writer in the *Engineering and Mining Journal*, vol. xc, No. 27, p. 1303 (Dec. 31, 1910); *Manufacturers' Record*, Jan. 19, 1910, and Aug. 11, 1911; the *Mexican Mining Journal*, vol. xiii, No. 3, p. 21 (Sept., 1911), and the *Mineral Industry*, vol. xix, p. 616 (1910).

From time to time, during the last 10 years, I have examined many localities in west Texas from which specimens of nitrate of soda and nitrate of potash were obtained, but I have yet to see a place which presents commercial possibilities of any moment whatsoever. I make this statement at this time to allay certain reports which have found some credence among those whose hopefulness has run away with their judgment. Rich specimens have been found at more than one locality, but in every case the situation has been such as to forbid any expectation of the finding of workable deposits.

Mention should be made of a locality on the Devil's River, Val Verde



FIG. 3.—EXPOSURE OF TRACHYTE CARRYING THIN SEAMS OF NITRATE OF SODA, NEAR CANDELARIA, PRESIDIO COUNTY, TEXAS.

County, described by J. A. Udden,⁵ which presents many of the characteristic features of so-called "deposits" of nitrates in Texas.

The saltpeter here was found in a cave on Devil's River and was formed by natural process from human excrements and blood waste. The place was an old camping ground and the accumulated rubbish, 136 ft. long and about 34 ft. wide, averages 5 ft. in thickness. In the uppermost foot there was 20 per cent. of saltpeter, in the second foot 7 per cent., in the third

⁵ Report on a geological survey of lands belonging to the New York & Texas Land Company, Ltd., in the Upper Rio Grande Embayment in Texas (Augustana Book Concern, Rock Island, Ill., 1907).

foot 1 per cent., in the fourth and fifth feet less than 1 per cent., while the sixth foot contained none.

The cave is located under an overhanging cliff of limestone about $\frac{5}{8}$ mile from where Indian Creek empties into the Devil's River. A little potash is found in certain bat caves in Texas, which have been partly burned, the potash appearing in the ashes, but the amount present is commercially negligible. The largest of these caves, containing, probably, several thousand tons of bat guano, is in Burnet County, about 25 miles northwest of the town of Burnet, and about 14 miles west of the railroad from Burnet to Lampasas.⁶

In summing up the entire matter, it can be said that the only hopeful outlook for the existence of workable beds of potash salts in Texas is in the direction already indicated by Dr. Udden, and in the almost wholly unknown region southeast of and bordering on New Mexico. Explorations in either of these areas would have to be in depth, with systematic chemical examination of cuttings and return waters.

⁶ Phillips, William B.: The Bat Guano Caves in Texas, *Mines and Minerals*, vol. xxi, No. 10, p. 440 (May, 1901).

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Mayari Iron-Ore Deposits, Cuba

BY J. F. KEMP, NEW YORK, N. Y.

(New York Meeting, February, 1915)

Introduction

THE *Bulletin* of the Institute for March, 1911, is chiefly devoted to papers upon the iron ores of northeastern Cuba. At that time information about the new developments in the peculiar brown hematites of the region was becoming widely circulated and interest was especially keen. For some years exploration had been conducted with pits and trenches and so great an area had been shown to contain ore in the Mayari, Moa, and Cubitas or San Felipe districts, that the reserves conveniently situated for the consumption of American furnaces were estimated as quite two billions of tons. In the subjoined footnote will be found a list of the principal papers of special interest which have already been published.¹ The further remark may, however, be made that the

¹ C. W. Hayes, T. W. Vaughan, and A. C. Spencer: Report on a Geological Reconnaissance of Cuba, under the Direction of Gen. Leonard Wood, Military Governor. The work in the Mayari, Moa, and Cubitas areas seems to have been chiefly done by Mr. Spencer, pp. 28, 83, 84 (1901).

Anon: The Mayari Iron-Ore District of Cuba. *The Iron Age*, vol. lxxx, pp. 421 to 426 (Aug. 15, 1907).

Anon: Iron Mining in Cuba. *The Iron Age*, vol. lxxxi, pp. 1149 to 1157 (Apr. 9, 1908).

A. C. Spencer: Three Deposits of Iron-Ore in Cuba. *Bulletin No. 340, U. S. Geological Survey*, pp. 318 to 329 (1908).

C. M. Weld: The Residual Brown Iron-Ores of Cuba. *Trans.*, xl, 299 to 312 (1909).

J. S. Cox, Jr.: Iron-Ores of the Moa District, Oriente Province, Island of Cuba. *Trans.*, xlii, 79 to 90 (1911).

W. L. Cumings and B. L. Miller: Characteristics and Origin of the Brown Iron-Ores of Camaguey and Moa, Cuba. *Trans.*, xlii, 116 to 137 (1911).

C. W. Hayes: The Mayari and Moa Iron-Ore Deposits of Cuba. *Trans.*, xlii, 109 to 115 (1911).

C. K. Leith and W. J. Mead: Origin of the Iron-Ores of Central and Northeastern Cuba. *Trans.*, xlii, 90 to 102 (1911).

J. E. Little: The Mayari Iron-Mines, Oriente Province, Island of Cuba, as Developed by the Spanish-American Iron Co. *Trans.*, xlii, 152 to 169 (1911).

A. C. Spencer: Occurrence, Origin and Characters of the Surficial Iron-Ores of Camaguey and Oriente Provinces, Cuba. *Trans.*, xlii, 103 to 109 (1911).

D. E. Woodbridge: Exploration of Cuban Iron-Ore Deposits. *Trans.*, xlii, 138 to 152 (1911).

nature of the ore has been the subject of an interesting litigation, because under the peculiar property rights inherited by the Cuban government from the previous Spanish authorities, a distinction is made between pigments and iron ores. The necessity therefore arose for deciding that these deposits were iron ore rather than raw materials for paint. The decisions of the Cuban courts have now interpreted them to be actually iron ores. The testimony of a number of geologists, specially qualified to speak authoritatively on this question, was sought for the judicial hearings and thus some of the earlier special studies, as printed in the *Bulletin* and *Transactions*, were made.

In summary, the published descriptions already impart to all readers the location of one great district near the town of Mayari, southeast of Nipe Bay, in northeastern Cuba; of a second, 40 miles and more farther east, called the Moa district; and of a third, about 160 miles to the northwest of Mayari, called the Cubitas in the papers of Cox and Spencer, but the San Felipe by Cumings and Miller. Still other areas of the red ferruginous soil are apparent in portions of western Cuba, but in most, if not in all, cases, they are so far from the sea, or so valuable for agriculture, as not to have attracted attention as a source of iron. Goodly percentages of iron are of course primarily necessary, coupled with absence of objectionable components, and so situated as to give transportation to American smelting centers at low freights. The chief interest at present is therefore attached to the Mayari district, as it is the only one as yet in active operation. In the Mayari district the ore is 15 miles from salt water, but when reached it forms a continuous area. In the Moa district, the ore-bearing ground even runs down to the sea-coast in places, but as a whole it is more dissected by erosion, so that the productive area is cut up into separate blocks. The San Felipe (or Cubitas) is farther from a deep harbor than either of the others. In all the three districts the ore mantles the tops of relatively elevated areas. In the Mayari it covers a plateau ranging about 1,800 ft. above tide. In the Moa district the ore near the coast is on the tops of hills of moderate altitude but rises to the south. In the San Felipe it covers a plateau at 450 to 500 ft. above the sea.

History

Crusts and concretionary masses of brown iron ore were early noted in northeastern Cuba. J. S. Cox, Jr., writing in 1911 (see citation, p. 79), states that claims were located upon them in the Moa district more than 20 years earlier. A. C. Spencer remarked them in 1901 as appearing in red clay, but no one seems to have realized until three or four years later that the entire mass was high enough in iron to be an ore. Under J. S. Cox, Jr., explorations were begun in the Mayari district in 1904 upon the crusts or "plancha," and then analyses revealed the fact that

not only the upper, dull-red portion of the so-called clay was valuable, but also the lower yellow parts as well. The construction of the Mayari plant of the Spanish-American Iron Co. was begun in 1907 and was completed in December, 1909. Shipments have been active ever since. The plant and mines were well described with maps and views by J. E. Little in 1911, as cited above. The chief changes since then have been the greater extent of the pits on the plateau at Woodfred, and the improvements in the village of Felton, the shipping port on Nipe Bay. The ore is principally treated at Sparrows Point, Md., and Steelton, Pa. It is shipped both in the crude state and as so-called "nodulized ore," or the product of kilns similar to modern cement kilns, in which the water, both absorbed and combined, is driven off and the fine ore is half fused or fretted into nodules. Mushiness in the stack is thereby prevented and the rather heavy percentage of water (25 per cent. absorbed and 14 per cent. combined) is driven off, to the diminution of freight charges. There or four per cent. of water is again absorbed in the cooling vats into which the kilns discharge. Analyses of the raw ore are customarily made on samples dried at or above 100° C. They have averaged by the year: Fe, 48 to 49; Ni, 1; SiO_2 , about 3; Cr, 1 to 2; Al_2O_3 , 11 to 11.5; combined H_2O , 13 to 13.5. The nodulized ore runs: Fe, 55 to 56; Ni, 1 to 1.2; SiO_2 , 4 to 4.4; Cr_2O_3 , slightly more than the raw ore; Al_2O_3 , 13 to 14; absorbed water, 3 to 3.5. In both, sulphur and phosphorus are negligible. In time, the great purity of these ores, combined with their percentages in nickel, and their convenient shipment from deep-water docks, should win a European as well as an American market.

Previous Work on the Geology

To the report of Messrs. Hayes, Vaughan, and Spencer, made in 1901 to Gen. Leonard Wood, at that time the Military Governor of Cuba, during the American occupation, we owe the only available systematic description of the island. As Messrs. Vaughan and Spencer were on the island but 12 weeks and Mr. Hayes but five, the report contains an extraordinary amount of detail when one recalls that the climate is tropical. The three geologists were chiefly busied with the mineral resources, but they give much information regarding general formations and made a careful summary of such previous literature as existed. Nevertheless, the report only whets our appetites for a full and accurate geological map and description of the island. Were it possible for the U. S. Geological Survey or some of our Academies of Sciences to make a co-operative survey with the Cuban government, as is now being done in Porto Rico in co-operation with the Porto Rican government, by the New York Academy of Sciences and affiliated New York City institutions, a great service would be rendered Cuba, as well as general science. Messrs.

Hayes and Spencer made a trip of eight days with a pack train from Santiago to Mayari and return. They crossed the tract of the Mayari mines and have drawn a geological section, which is here reproduced from their Plate XVIII, Section 6, in so far as it relates to the Sierra Nipe, on which the ore lies (Fig. 1).

The following quotations are taken from the report:

P. 28. "Along the line of the trail which crosses the Sierra Nipe leading from San Luis to Mayari, near the line of the section presented in Plate XVIII, Fig. 6, there is a belt of metamorphic and dense igneous rocks, between the Oligocene limestone and the serpentine which forms the axis of the Sierra. This series is very poorly exposed, but is probably older than the serpentine. Indications of a similar series were observed near the edge of the serpentine on the south side of the Sierra Cristal in the vicinity of Mayari Arriba.

"The main mass of the Sierra Nipe is composed of serpentine, which is shown by the microscope to have originated through the alteration of a rock originally composed largely of bronzite, fragments of which mineral are still frequently to be observed in the otherwise completely altered mass. Since this chemical alteration occurred, the serpentine has been intruded by dense diabase. These last-named rocks occur in



1, Metamorphic rocks; 2, Serpentine with diabase and gabbro intrusions; 7, Massive limestone (Oligocene?); 8, Coastal soborrucos and Oligocene marls.

FIG. 1.—REPRODUCTION OF NORTHERN PART OF GEOLOGICAL SECTION, PLATE XVIII, FIG. 6, OF REPORT BY HAYES AND SPENCER.

such abundance that it is impossible to find any large area of the serpentine which is not cut at least by small dikes of the later rock, and frequently it is found in very large masses.

"On the northern side of the Sierra Nipe the Oligocene limestone reaches an elevation of about 900 ft., dipping northward toward the sea. In the higher exposures the dips are between 10° and 15°, but become less as the coast is approached."

On pp. 83 to 84 the following remarks appear with reference to the shot-like masses and crusts of limonite now called *plancha*:

"Occupying the general region between Nipe Bay and Moa Bay, and somewhat back from the northern coast, there is a region reaching a general elevation of from 1,500 to 2,000 ft., and occupied by serpentine and other igneous rocks. Upon the top of this sierra there are many large areas which are practically level, and these are always covered by a thick mantle of red clay, which contains a large proportion of iron ore in the form of spherical pellets. Locally this material entirely replaces the clay, and the separate particles are cemented together by ferruginous materials, making a spongy mass of brown iron ore. Similar occurrences of shot and massive ore were noted upon the tops of certain hills lying to the north of the city of Puerto Principe, and following the general trend of the Sierra Cubitas. The rock in this vicinity is also serpentine, and the ores have identical characteristics with those of the region mentioned above. Analyses were made of these residual ores collected near Rio Seco along the trail between Mayari and San Luis."

The following two analyses are given with two others from localities of no interest in this connection:

	1 Per Cent.	2 Per Cent.
Moisture.....		0.56
Iron.....	52.00	54.69
Manganese.....	0.364	0.594
Phosphorus.....	0.0368	0.0189
Silica.....	2.62	2.51
Chromium.....	tr.	present
Titanium.....	0.25	

1. Iron ore from Sierra Nipe near trail crossing the Rio Naranjo, about 10 miles from Mayari, Santiago Province (now Oriente Province).

2. Iron ore from Sierra Nipe near Rio Seco, Santiago Province (now Oriente Province).

"These residual ores are locally known as 'tierra de perdigones' or 'moco de herrero,' signifying shot-soil and blacksmith's waste, either of which terms is a very apt designation. Rodriguez Ferrer is authority for the statement that hydrated oxide of iron in the form of pellets in the soil occurs at various points in the Island. The following localities are mentioned: Province of Pinar del Rio, between Consolacion del Sur and Candelaria; Matanzas Province, in the Sierra Morena, between Cardenas and Sagua la Grande; Loma Iman near the city of Puerto Principe, and Monte Libano north of Guantanamo, Santiago Province. The amount of these ores in various parts of the island is certainly very large and it seems not improbable that they may eventually find a market in the United States in cases where they are situated near a sufficient supply of running water for washing them free from the clay with which they are mixed."

That the mantle of surface materials was due to the alteration or weathering of serpentine was recognized by Hayes and Spencer, and some notes on the mother rock of the serpentine were made, based on microscopic study. They also noted the later dikes described as diabase, presumably observed in ascending and descending the plateau, since on the top everything was then concealed. So far as the ore is concerned it only remained for Mr. Cox and his engineers to discover that the "clay" was also iron ore.

All the other observers corroborate the derivation of the ore by the weathering of serpentine, and Mr. Spencer in his paper of 1908 goes more at length into this subject, and notes the parallels with the ores formerly dug on Staten Island, N. Y., and with the deposits known to exist at Clealum, Wash. C. M. Weld in 1909 takes up the process of tropical weathering much more fully and acutely recognizes the similarity of the ore to the laterites of India. Mr. Weld also gives four fairly complete analyses from Mayari, Moa, Taco, and Navas, which he had evidently recast into their possible minerals because he states, p. 302: "Within the clay-ore are found disseminated nodules and pellets of brown ore ranging apparently through all the hydrated forms from limonite to turgite; hematite also is present and at times magnetite." Mr. Weld

obviously recognized that the combined water was too small to satisfy the alumina for bauxite ($\text{Al}_2\text{O}_3 \cdot 2\text{H}_2\text{O}$) and the iron for limonite ($2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$). The same point is emphasized by C. K. Leith and W. J. Mead and will be more fully discussed later on. Mr. Weld gives the first published analysis of the serpentine, placing by its side a reconstructed complete analysis of the ore. The serpentine came from Moa Bay. It is of special interest to compare these analyses with a similar analysis from Mayari later given. In quoting Mr. Weld's analyses, the components have been rearranged to correspond with the order now almost universal in rock analyses.

	Serpentine, Moa Bay	Iron Ore, Moa
SiO_2	37.29	1.71
TiO_2	See note	0.14
Al_2O_3	1.33	11.60
Fe_2O_3		66.90
Cr_2O_3	See note	2.65
FeO	8.55	
MnO	tr.	0.80
NiO	See note	0.60
MgO	36.53	See note
CaO	0.29	See note
K_2O	tr.	See note
Na_2O	0.39	See note
H_2O	15.27	13.15
P_2O_5	0.07	0.07
	99.72	97.62

NOTE.—Under the first analysis, 0.28, the balance to 100 per cent., is assigned to TiO_2 , Cr_2O_3 , and NiO . Under the second analysis, 2.38, the balance to 100 per cent., is ascribed to MgO , CaO , K_2O , and Na_2O .

W. L. Cumings and B. L. Miller specially studied the field north of Camaguey, which their predecessors had apparently named the Cubitas, but which, because of its separated position from the Cubitas Mountains proper, they prefer to call San Felipe. They also add interesting and important details on the Moa field. The San Felipe ores contain in their lower portions impressive amounts of chert, which fails in the other districts. They note large, recognizable crystals of pyroxene in the serpentine, and remark the relations of the serpentine with massive Triassic limestone (p. 119 of their paper) along the Jigüey River. Hayes, Vaughan, and Spencer do not mention Triassic rocks in their stratigraphic column (p. 33 of their report to General Wood) and it would be of great interest to record the evidence. Messrs. Cumings and Miller were well aware that all the iron in the ore was not hydrated, and mention both magnetite and hematite (p. 128) as components. Small pellets of hematite and some magnetite were detected in microscopic slides of serpen-

time (p. 135), so that these minerals are regarded as survivors from the serpentine.

C. K. Leith and W. J. Mead visited all three of the large districts but gave special attention to the Mayari. They emphasize the presence of bauxite ($\text{Al}_2\text{O}_3 \cdot 2\text{H}_2\text{O}$) and gibbsite ($\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$) in the ore. Eleven samples were taken in a 10-m. section of the ore and were analyzed, as was also the bedrock serpentine, so as to trace out the changes. The

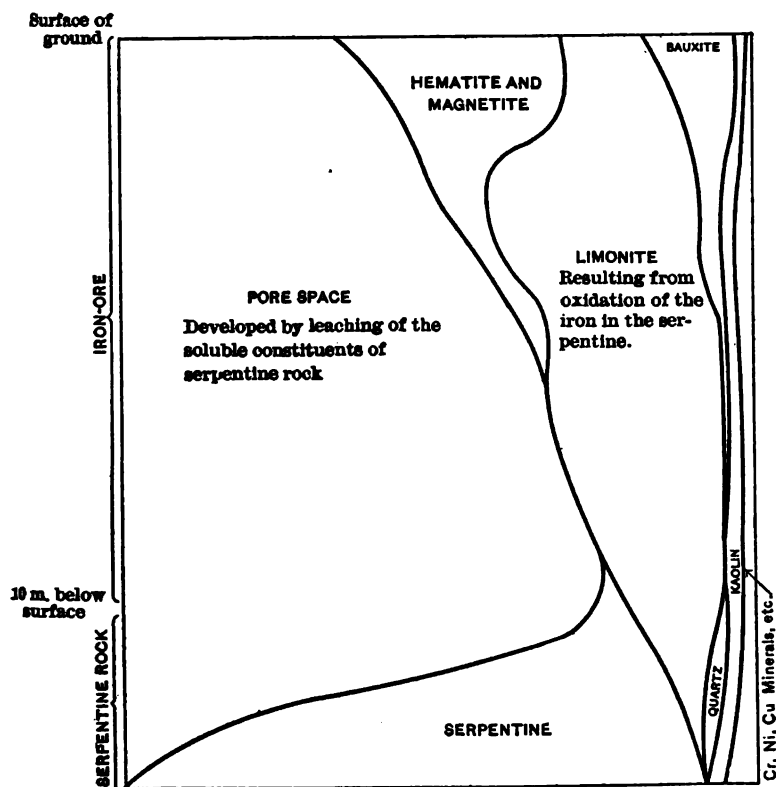


FIG. 2.—DIAGRAM BY C. K. LEITH AND W. J. MEAD ILLUSTRATING THE PHYSICAL AND MINERALOGICAL CHANGES OF THE SERPENTINE IN ITS PASSAGE TO IRON ORE. REPRODUCED FROM P. 93 OF VOL. XLII OF THE *Transactions*.¹

analyses are not published, but a chart, plotted so as to show the pore space, progressively left by the lost components, and the relative volumes occupied by the residual minerals, is given and is here reproduced (Fig. 2). In order to run some comparisons with the mineralogy of the ore as determined by recasting some new analyses, the writer has scaled off in fiftieths of an inch the cross-section of the chart, first at the top, next in the middle of the ore, and lastly at the bottom of the ore. After leaving out the pore space the other proportionate measurements have

been multiplied by the respective specific gravities of the minerals involved, and the products have been recalculated to percentages with a total of 100, so as to express the percentages of the minerals by weight. These will be used in comparison later on. The general chemistry and physical changes in the passage from serpentine to ore are carefully reviewed by the two authors. On the geology the following sentences are significant:

"Geologists are substantially agreed that the ores of the Moa and Mayari fields are residual or mantle deposits resulting from surface alterations in place, of serpentine rock, which in turn probably represents the alteration of some other rock like a peridotite not yet disclosed by underground explorations. With the serpentine there are present very minor quantities of intrusive dike rocks, high in alumina, which by the surface alterations yield clay, not iron ore."

This summarized review will afford, the writer hopes, a serviceable idea of what has been already recorded. Some further notes on the general geology and some petrographic details of the serpentine will be next given and then the actual and so far as possible the proportionate mineralogy of the ore will be further discussed.

Geology

At the coast and along the low bluff arising from the water, a marly clay with lumps of limestone intermingled forms the surface. The same general formation extends for some miles back, and appears as thin, flat beds of tender limestone in thicker beds of clay or marl. Twelve miles from the coast, quite solid, white limestone appears in flat beds and is quarried as shown in Fig. 3. No fossils were found after careful search, nor could the writer learn of any from the men at work. Presumably the same formation extends to the plateau and appears as far up as the head of the lower incline. It could also be seen across a valley, as shown in Fig. 4, a photograph taken from the head of the lower incline looking westward. The point of view was at 680 ft. elevation above the sea. Above this point serpentine soon appears and, with the associated dikes of diorite as later described, constitutes the plateau so far as known. Undoubtedly more detailed exploration of the valleys, and of the lower contact of igneous rocks on limestones would bring to light additional facts of much interest. There must be a vertical thickness of exposed serpentine of at least 1,200 ft.

The cuts of the upper incline give an excellent section of the serpentine, since they have been sunk as much as 40 ft. below the surface. Good, fresh rock has thus been afforded for study. Other cuts along the railway between the head of the incline and the station at Woodfred give good exposures, and a third cut made in the bedrock near the southwest workings, known as the Three-Hill mine, has opened up excel-



FIG. 3.—LIMESTONE QUARRY AT ARROYO SEBORUCO, 12 MILES FROM FELTON.



FIG. 4.—VIEW OF LIMESTONE FOOT HILL OF SERPENTINE PLATEAU, LOOKING WEST FROM HEAD OF LOWER INCLINE.



FIG. 5.—LOWER TWO-THIRDS, BASTITE (HIGH RELIEF) AND SERPENTINE (LOW RELIEF) FROM PYROXENE; UPPER THIRD, OLIVINE CHANGING TO SERPENTINE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.



FIG. 6.—THE SAME AS FIG. 5, BUT TAKEN WITH CROSSED NICOLS.

lent material for microscopic study. A very fresh and surprisingly unaltered boulder of rock was met in the mines at the Y, where the spur to the northeast branched from the main line, in 1914.

The serpentine is derived from a peridotite which consisted of olivine, and pyroxene, apparently an orthorhombic variety which is now thoroughly changed to bastite and fibrous serpentine. The original rock probably also contained monoclinic pyroxene, since in the slides some extinctions not parallel with the cleavage would suggest it. The chemical change has been so thorough as to destroy most of the optical properties of the original. The bastite has feeble double refraction, giving



FIG. 7.—BASTITE PRESERVING CLEAVAGES OF A PYROXENE CRYSTAL IN RIGHT CENTER. ABOVE IT A DARK PICOTITE. GENERAL MASS SERPENTINE FROM OLIVINE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.

pale grays. It still displays prismatic cleavage which has parallel extinction. It passes into fibrous serpentine, as is illustrated in the lower two-thirds of Fig. 5. The serpentine fibers maintain parallelism with the prismatic cleavage of the original pyroxene. The fibers are curved slightly in Fig. 5, suggesting some pressure effects. In the change to serpentine some highly refracting mineral has developed in the cracks, which behaves like quartz, but extinguishes at various angles across the elongation. It is shown in Fig. 6, which is the same field as Fig. 5. In Fig. 7 is a pyroxene crystal with its prismatic cleavages well shown and in addition pronounced partings parallel with the vertical pinacoids.

The dark, angular grain just above the pyroxene crystal is a brown, isotropic mineral of high index and is probably the chrome spinel, picotite. It is quite frequently seen in the slides and is believed to be picotite because the dark brown mineral becomes transparent too readily to be chromite. The slides show occasional opaque metallic grains, which are doubtless magnetite and chromite. The serpentine develops in the olivine in cross fibers, along cracks, and is well illustrated in Fig. 8. A further stage, from which the olivine has almost all disappeared and serpentine with the so-called mesh structure has taken its place, is illustrated in Fig. 9. Numerous black specks of secondary iron ore are shown.



FIG. 8.—SHOWING GENERAL DERIVATION OF SERPENTINE FROM OLIVINE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.

Finally, in Fig. 10, we have only serpentine left, with the mesh structure emphasized by taking the photograph with crossed nicols.

In the cuts in the serpentine along the upper incline rather poorly defined streaks of dense black or dark-gray rock can be frequently detected. So much movement or crushing has taken place, either from dynamic disturbance or from the swelling and adjustment incident to the change from peridotite to serpentine, that their regularity is greatly obscured. Doubtless they were once dikes, but now on the edges they shade into the schistose serpentine and probably have themselves altered to it in part. Under the microscope they are all diorites with strong suggestions of original diabasic texture. The long banded rods in Fig.

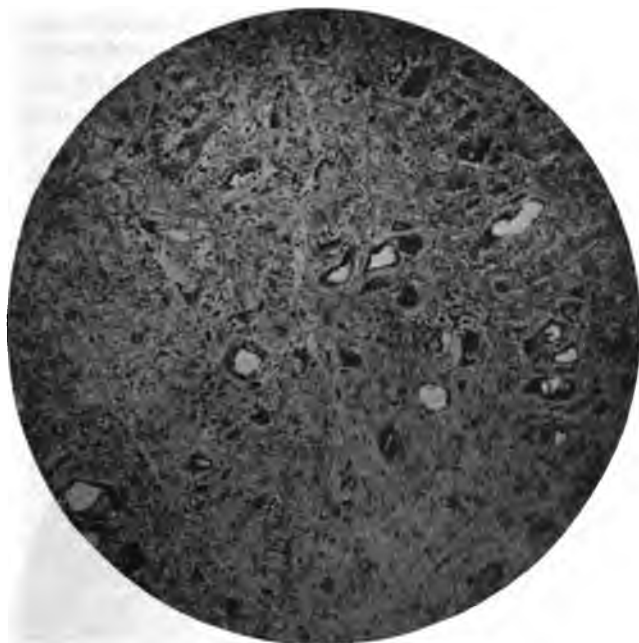


FIG. 9.—SHOWING ALMOST COMPLETE CHANGE OF SILICATES TO SERPENTINE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.



FIG. 10.—SIMILAR VIEW TO FIG. 9, BUT TAKEN WITH CROSSED NICOLS TO BRING OUT THE MESH STRUCTURE OF THE SERPENTINE.

11 are the plagioclase. The other minerals of uniform gray to black tints are hornblende. Magnetite in small grains can also be detected in the slides. In other dikes the hornblende becomes more abundant and the plagioclase less. In Fig. 12, taken in white light, the darker mineral making up three-quarters of the field is all hornblende. The lighter areas are plagioclase. Some dead-black magnetite is also shown. The boulder mentioned above as found near the Y of the mine railway is a coarser diorite, rich in feldspar, which even shows micropegmatitic structure. Although appearing as a loose boulder, this rock proved to be surprisingly unaltered in the thin section, all the minerals being clear



FIG. 11.—DIORITE WITH DIABASIC TEXTURE. THE DARK AND LIGHT Banded MINERAL IS PLAGIOCLASE; THE DARK MINERAL IS HORNBLENDE. Crossed NICOLS. ACTUAL FIELD, 0.1 IN.

and fresh. It must indicate a dioritic dike in the serpentine. Mr. Spencer has reported diabase dikes to be numerous and widespread in the serpentine, and, as he crossed this area years before mining was begun, he must have observed them along the steep sides of the plateau, since practically no outcrops appear on top. One cannot but suspect from the textures of the dikes here illustrated that the hornblende is secondary after augite, and the slides justify this surmise. A few cases of unchanged augite have been observed in the midst of the hornblende, whose leek-green color would almost of itself be demonstrative of a secondary variety. In slides made from several dikes gathered on the

incline the feldspar has passed to an aggregate of not strongly bi-refracting rods, which are probably some zeolite. Doubtless this is the first stage in the alteration, which in the long run yields laterite. Although no chemical analyses have been made, all these diorites will run much higher in alumina than the serpentine. They will certainly yield from 15 to 18 per cent., and are doubtless responsible for some of the alumina in the ore and its local exceptional abundance.

An analysis of the fresh serpentine from the Three Hill mine cut, from which Figs. 5, 6, and 7 were taken, has been kindly made by T. C. Kraemer, chemist of the Spanish-American Iron Co. at Felton. By its side is placed the analysis given by C. M. Weld, as earlier noted:

	I Serpentine, Woodfred	II Serpentine, Moa
SiO ₂	37.28	37.29
Al ₂ O ₃	2.45	1.33
Fe ₂ O ₃	5.14	
Cr ₂ O ₃	1.68	
FeO.....	5.14	8.55
MnO.....		tr.
NiO.....	0.30	
MgO.....	34.59	36.53
CaO.....	none	0.29
K ₂ O.....		tr.
Na ₂ O.....		0.39
H ₂ O (combined).....	12.80	15.27
H ₂ O (absorbed).....	0.91	
P ₂ O ₅	0.013	0.07
	100.303	99.72

In analysis II, Mr. Weld assigns to the difference between the total and 100, TiO₂, Cr₂O₃, and NiO, amounting to 0.28 per cent.

Much interest attaches to the recasting of these analyses into the percentage composition of the component minerals of the rocks. At first sight the presence of so much Fe₂O₃ in the analysis of the Woodfred sample strikes one as unusual, since in general in serpentine the iron oxide is chiefly in the state of protoxide. If, however, in recasting we assume it to be ferrous, we have far too little silica to satisfy the serpentine molecule 3(MgO.FeO), 2H₂O, 2SiO₂. The ferric oxide has therefore been arbitrarily distributed between limonite and magnetite, the governing motive being the exhaustion of the silica. As checked by study of thin sections, we are thus not very far from the true components of the rock nor their relative amounts. The chief variation lies in the undoubted presence of some unaltered olivine. Bastite is generally credited with the same composition as serpentine. Mr. Weld's analysis is stated to be the average of a number of samples. When one attempts to recast it, assigning the magnesia and the ferrous iron to the serpentine molecule, there is lack of silica, and the difficulties are increased by assuming any

probable silicate for the soda and lime. Any one of the averaged individual analyses would be better for recasting than the average of them all.

Fully realizing, therefore, that the percentages given by the recasting of the Woodfred sample are approximations rather than mathematically correct expressions, we have some idea of the minerals or compounds whose alteration has produced the ore. Sharper definition is thus given to our conceptions of the necessary changes than if we only had the analysis before us. The total does not exactly check with the analysis, from small rejections of decimals, here and there.



FIG. 12.—MORE BASIC DIORITE WITH PREDOMINATING HORNBLENDE. ACTUAL FIELD, 0.1 IN.; WHITE LIGHT.

	Serpentine, Woodfred		Total
Magnesia serpentine.....	79.33	}	85.986
Iron serpentine.....	6.174		
Nickel serpentine.....	0.482		
Kaolinite.....	1.548		
Bauxite.....	2.346		
Limonite.....	4.114		
Magnetite.....	2.320		
Chromite.....	2.472		
Unassigned water.....	0.396		
Absorbed water.....	0.910		
P ₂ O ₅	0.013		
	100.105		

The change to ore obviously involves the entire disappearance of the magnesia; the nearly total removal of the silica; the probable change of the nickel-serpentine to one or more of the other hydrated silicates such as have been named from various nickel mines—genthite, connarite, garnierite, etc., and of which F. W. Clarke² states: "These silicates are rarely if ever found as definite mineral species, although they have been described as such." One called nepouite has, however, the same formula as nickel-serpentine, $3(\text{NiMg})\text{O}$, $2\text{SiO}_2 \cdot 2\text{H}_2\text{O}$, or $(\text{NiMg})_2\text{Si}_2\text{O}_7 \cdot 2\text{H}_2\text{O}$ as cited by Dr. Clarke. In the passage to ore, whose composition is later shown, some nickel surely migrates downward, because it is not increased in percentage in the same ratio as the alumina, and the serpentine below the ore is much richer in nickel than the sample above analyzed. Instead of 0.3 per cent. we may find 2 per cent. and over of NiO . The kaolinite breaks up in large degree to bauxite ($\text{Al}_2\text{O}_3 \cdot 2\text{H}_2\text{O}$) or some other aluminum hydrate such as gibbsite (called also hydrargillite), $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$. There seems to be no good reason why there should not be also present $2\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, corresponding with limonite, but no such mineral has yet been named. Even bauxite itself may be interpreted as an intimate mixture of diasporite, $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$, and gibbsite, as Dr. F. W. Clarke remarks (p. 472), since the last two are known in a crystallized condition while bauxite is not. The chromite and magnetite survive with little change, both being very resistant minerals. Hematite may also be a mineral in the serpentine, as considered by Cumings and Miller, even though not calculated as such in the above recasting. It may survive in the residual product. The iron passes into some form of hydrated oxide, as will be later shown. Limonite, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, probably is in excess over goethite, $\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$, and turgite, $2\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$. Some iron doubtless has also escaped. All these changes are facilitated by the situation of the serpentine on a high plateau, whose top is 1,500 to 1,900 ft. above the surrounding lower country, and which has extremely steep sides. Underground discharge for the percolating rain water has been extremely free, since serpentine is pre-eminent among rocks for checks, slips, and cracks. The great increase in volume incident to the combination of so much water with the anhydrous silicates of the peridotite makes readjustments on a grand scale inevitable.

The Ore

In the faces of the pits as now extensively exposed, the observer can readily note that there are three distinct layers: An upper, of crimson-brown hue; a middle one, yellowish brown, and a bottom layer of a lighter shade of yellowish brown. In one face, called the Three-Hill cut, the

² F. W. Clarke: Data of Geochemistry, *Bulletin No. 491, U. S. Geological Survey*, p. 664 (1911).

upper was noted at 5 to 6 ft.; the middle at 6 to 12 ft.; and the lower at 4 to 6 ft. The two upper layers are illustrated in Fig. 13. Analyses of each layer, but from another station, will be subsequently given. As distinct from these varieties the engineers in charge have noted that in the occasional and rather rare spots in the residual mantle where the iron percentage is too low for mining there appears at the surface a peculiar purple color, quite easily recognized and an indication of high alumina in the samples. The color appears to be due to the relatively rich admixture of normally white bauxite with the darker-hued brown iron ore. The explanation of the higher alumina is to be found in local changes in the original rock, as later set forth.



FIG. 13.—FACE LEFT BY STEAM SHOVELS IN THREE-HILL MINE, AND SHOWING TOP, DARK LAYER, AND MIDDLE, LIGHTER LAYER. THE TOP DARK LAYER HAS CAVED DOWN ON THE BASE, CONCEALING BOTTOM LAYER.

In some places, at the surface or a few feet below it, slabs and even continuous sheets of solid iron hydrate appear and afford the cellular varieties of brown ore, very similar to the crusts and lumps long familiar in the mines of the Appalachian belt. As earlier stated, the solid ore is called *plancha*. In the residual mantle shots and larger lumps of solid brown ore are at times intermingled, chiefly in the upper, darker layer. The general run of the ore is, however, earthy, and reminiscent in the strongest degree, alike in color and texture, of the Mesabi ores. The higher con-

tent of absorbed water, the higher alumina, and the lower silica of the Mayari ores give them perhaps a somewhat more spongy aspect than one notes on the Mesabi range; on the other hand, at Woodfred there is no overburden whatsoever, and the ore is obtained from grassroots to bedrock.

The recently mined ore has a peculiar mealy character, reminding one of nothing so much as dampened meal, but as it dries out this character disappears. No doubt the colloid nature of the hydrates of alumina and iron is the cause of the peculiarity.

The following analyses made by T. C. Kraemer, chemist of the Spanish-American Iron Co., were based upon samples gathered by the writer to illustrate the three contrasted layers. The samples were taken as nearly as practicable in a vertical section.

	I Surface Ore. Sta. 11,35,10. Crimson-brown	II Middle Layer. Sta. 11,35,10+15 Yellow	III Bottom Layer. Sta. 11,35,11 Yellow
SiO ₂	2.26	2.70	7.54
Al ₂ O ₃	14.90	7.13	4.97
Fe ₂ O ₃	68.75	71.89	64.81
Cr ₂ O ₃	1.89	3.17	3.66
FeO.....	0.77	1.29	1.49
NiO.....	0.74	1.60	2.75
MgO.....			1.50
H ₂ O combined....	11.15	12.90	12.75
Total.....	100.46	100.68	99.47
Metallic iron.....	48.65	51.32	46.52
Metallic nickel....	0.59	1.20	2.10
H ₂ O absorbed*....	4.62	9.72	27.00

* Determinations of absorbed water based on original sample. Other determinations on dried sample at 110° C.

From these analyses we draw the following conclusions:

Silica progressively increases from above downward. It is relatively high in III because of bits of included serpentine and probably because of the descent of the silica in solution and loss in largest proportion from the uppermost layer, which has been longest exposed to alteration.

Alumina decreases downward because it is the least soluble of all the components and therefore reaches a maximum in the most weathered portion.

Ferric oxide, as we pass downward, increases and then declines. In III the silica and magnesia of the bits of serpentine relatively reduce it. Doubtless, in the long run descending rain waters dissolve some iron from the upper layer and re-deposit it in part in the yellow middle layer. Some probably runs off altogether.

Chromic oxide progressively increases downward, a relation at first causing surprise. Since chromite is an extremely resistant mineral,

one would naturally expect it to become concentrated near the top. It must either work its way downward because of its relatively high specific gravity or else it must appreciably yield to waters in the long run. The former supposition is more probable.

Ferrous oxide impresses one at first sight as being sympathetic with chromic oxide, although a little below the requirements of chromite. The difference could be explained by small percentages of magnesia or even of nickel oxide in the chromite. Tests with the magnet, however, prove the presence of appreciable quantities of magnetite. It therefore seems more reasonable to assign the FeO to magnetite; and to infer from the insoluble nature of chromite that its FeO was finally determined as Fe_2O_3 . In recasting the analysis all the FeO has been used for magnetite, and enough Fe_2O_3 to furnish the necessary FeO for chromite has been taken for this purpose.

Nickel oxide progressively increases downward. It may be in very small part in the chromite. It is undoubtedly present as some hydrated nickel silicate. It is probably affected by the rain waters and has been removed by them to a relatively great degree from the upper layers. Concentration in the bedrock and in the bottom layers of ore is quite certainly shown by analyses, additional to the three cited.

Magnesia practically disappears with the general breaking down of the serpentine.

Combined water shows a slight increase, because surviving serpentine in the bottom layer accounts for some, and increased percentages of iron, presumably in the molecule, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, in the middle layer for the rest. There is reason to infer the increased percentage of magnetite and perhaps hematite in the top layer. Leith and Mead plot these two as forming more than half the solids by volume, and this would involve an even greater percentage by weight, because the minerals bauxite and kaolinite of low specific gravity enter in an important way into the residue. The writer's tests with the magnet and by microscopic study of the samples gathered for the analysis here published showed that the shots of ore were magnetic to an appreciable degree, perhaps 5 to 10 per cent. of them, but the fine ore was practically inert. The shots were all discolored with brown ore and when crushed showed a goodly portion of limonite in the powder.

If we seek to combine the iron and all the water of I into the common limonite molecule, there is too little water; in II there is a slight excess of water; and in III there is a large excess. But the alumina in each calls for some water, and it is therefore clear that all the iron is not in the limonite molecule, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, but there must also be present some iron hydrate calling for less water. From microscopic study, we are certain that there is a little magnetite, and this helps to solve the difficulty, but there must also be present large proportions of goethite, $\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$, and

there may be some turgite, $2\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$. The absorbed water is exceptionally low in I and II, and the small percentage is undoubtedly due to evaporation during a period of slight rainfall.

Much interest attaches to the recasting of these analyses in the endeavor to bring out the actual or probable proportions in which the constituent minerals enter and the identity of the minerals themselves. The low percentage of silica makes clear that kaolinite, $2\text{H}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$, one of the first minerals to be considered, can be present in only small amounts. The presence of chromic oxide makes justifiable and indeed unavoidable the conclusion that chromite, $\text{FeO} \cdot \text{Cr}_2\text{O}_3$, is also present. Recasting shows that there is from a little to a good deal too much Cr_2O_3 in the analyses for the reported FeO . As stated above, we infer from this, since chromite is a very insoluble mineral, that the reported FeO is in magnetite. On this assumption the magnetite has been calculated, assigning to it all the reported FeO , and then taking from the Fe_2O_3 enough to correspond to the FeO required for chromite. The resulting percentages are so close to the general indications of microscopic study as to be worthy of much confidence. We infer that the greater part of the nickel is combined with silica and water as some form of hydrated silicate. The normal nickel-serpentine molecule has been used. Leaving one side the nickel as one of the minor components, we recognize the fact that the iron hydrates and the alumina hydrates are the large components.

As a further preliminary to the results of attempted recasting, a brief summary may be given of a careful microscopic examination made by Dr. Charles P. Berkey for the Spanish-American Iron Co., in order to determine the condition of the iron minerals and their relations to other components. The ore proved to be in largest part an extremely intimate mixture of some iron hydrate, assumed to be limonite; and some alumina hydrate, assumed to be bauxite; with minor proportions of recognizable chromite and admixtures in minute particles of quartz, epidote, hornblende, and possibly feldspar. By fractional treatment with hydrochloric acid, and study of residues, the above conclusions were reached. The sample was different from the ones used in the three analyses here published, but the results give us a definite and interesting point of view. Chamosite and thuringite, the hydrated silicates of iron, familiar in the minette ores of Luxemburg and Lorraine, were considered, but no reason appeared for inferring their presence.

In the tables of recast results, different assumptions are made and tabulated for iron and alumina hydrates in the endeavor to work out combinations of the water, which is too small in amount for limonite, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, and bauxite, $\text{Al}_2\text{O}_3 \cdot 2\text{H}_2\text{O}$, alone. The methods employed are those now widely used for rock analyses.³

³ Kemp: *Handbook of Rocks*, pp. 161 to 179. Cross and others: *The Quantitative System for the Classification of Igneous Rocks* (Chicago, 1903). Finlay, G. I.: *Igneous Rocks* (New York, 1913).

	I Top Layer, Recast			II Middle Layer	III Bottom Layer
	Ia	Ib	Ic		
Limonite.....	39.66	21.32		59.09	69.96
Hematite.....	31.68				
Turgite.....		50.02	30.76		
Goethite.....			40.58	18.87	
Magnetite.....	2.55	2.55	2.55	4.18	4.87
Chromite.....	2.79	2.79	2.79	4.85	5.58
Bauxite.....	17.98	17.98	17.98	7.45	1.34
Kaolinite.....	4.05	4.05	4.05	3.78	9.80
Ni-serpentine.....	1.30	1.30	1.30	2.69	4.70
Mg-serpentine.....					3.45
Excess water.....					0.04
	100.01	100.01	100.01	100.91	99.74

In recasting analysis I, the relatively small amount of water has compelled three assumptions. In the first, after providing for bauxite, kaolinite, and nickel-serpentine, all the water is assigned to the limonite molecule and the excess of ferric iron is calculated as hematite. So much hematite is, however, contradicted by microscopic study of the sample. The magnetite and chromite account for approximately as much as one actually observes of the iron minerals which do not afford on crushing a yellow or brown powder. Therefore, a second recasting was carried out by allotting the water to limonite, $2\text{Fe}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$, and turgite, $2\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$. By this assumption yellow-brown hydrates of iron alone are used. The third recasting was made also to use only iron hydrates; but assuming turgite, $2\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$, and goethite, $\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$, one sees that hematite is not a necessity in any great quantity. In the diagram given by Professors Leith and Mead, if we select sections at the top, middle, and near the bottom of the ore, scale off a proportionate part for each mineral by its specific gravity, and then reduce the products to percentages, we will turn the diagram into percentages by weight. In doing this the following specific gravities have been used: Magnetite and hematite, 5; limonite, 4; bauxite, 2.4 (as given by J. C. Branner, *Journal of Geology*, vol. v, p. 270, 1897); kaolinite, 2.5; chromium, nickel, and copper minerals, 4.5 (a value probably high); serpentine, 2.6.

	I Top	II Middle	III Bottom
Hematite and magnetite....	64.8	15.0	
Limonite.....	19.2	72.4	75.4
Bauxite.....	12.7	2.4	
Kaolinite.....	1.6	3.0	8.4
Chromite, etc.....	1.7	7.2	9.0
Serpentine.....			7.2
	100.0	100.0	100.0

In the smaller components, kaolinite, chromite, and serpentine, the results are generally reasonably parallel with those obtained by recasting the three analyses given above. The bauxite is not beyond the allowable differences of samples. The chief contrasts are in the distribution of the iron oxides among the several iron minerals in the top layer. The samples used in the present paper indicate less hematite and magnetite.

Samples were also collected at station 21.02.16 of the lean so-called ore, of purple color, and too low in iron for mining. A second sample of a cellular decomposed mass, supposed at the time to be bedrock, was taken from the base of the exposed section. Both were kindly analyzed by T. C. Kraemer. Apparently the samples were derived from the weathering of the peridotite, where penetrated by diorite dikes.

	I Upper Portion	II Lower Portion
SiO ₂	6.88	2.28
Al ₂ O ₃	23.44	39.80
Fe ₂ O ₃	54.46	34.44
Cr ₂ O ₃	1.89	0.27
FeO.....		
NiO.....	0.25	0.09
H ₂ O combined.....	13.84	24.01
	100.76	100.89
H ₂ O absorbed.....	4.72	1.86

	I Upper Portion	II Lower Portion
Limonite.....	38.17	40.06
Hematite.....	20.80	
Bauxite.....	24.02	19.05
Gibbsite.....		36.19
Kaolinite.....	14.49	4.90
Ni-serpentine.....	0.41	
Chromite.....	2.85	0.43
NiO not used.....		0.09
	100.74	100.72

In No. I we are forced to use hematite, or appeal again to goethite and turgite, which latter step in this case has not been taken. In No. II the water is so high as to necessitate the presence of gibbsite, Al₂O₃.3H₂O, along with bauxite. The gibbsite molecule is important in the laterites. Undoubtedly in all these processes of weathering colloids are formed, since, with the exception of chromite, hematite, and in the previous analyses magnetite, practically all the molecules employed appear in amorphous and structureless forms, which are most reasonably explained by assuming an original colloid. At the International Geological Congress, held in Toronto, in August, 1913, Prof. Paul Krusch, of Berlin, emphasized the importance of this form of matter both in the results of surface weath-

ering and in a few possible situations placed deeper within the earth.⁴ The hydrated nickel silicates received special mention and were interpreted as colloidal precipitates.

Recently Described Areas of Laterite Ores

In the earlier papers cited on the Mayari and related districts mention is made of similar ores at Staten Island, N. Y.,⁵ and Clealum, Wash., and of others in the Mediterranean region. Besides these two, additional instances have recently been described to which reference may be made in conclusion. Prof. Alfred Lacroix, of Paris, has been for over 15 years carrying on extended studies of the tropical alteration of rocks in French Guinea in latitude 10 N.⁶ Professor Lacroix defines laterites (p. 259) as "The products of the decomposition of all rocks containing silicates of alumina, and which, from the chemical point of view, are characterized by the predominance of the hydroxides of alumina and iron generally with the oxide of titanium, after the elimination more or less complete of the other elements of the fresh rock: alkalies, lime, magnesia, and silica." While the original peridotite of the Mayari iron ore is one of the poorest of all igneous rocks in alumina, yet there was enough in it so that the residual products of its decay became relatively high in this oxide. Professor Lacroix gives (p. 296) the accompanying analyses, which range from much less altered peridotite than we have yet obtained at Mayari to iron ore apparently like the crusts of Cuban plancha.

Analysis (d) is analogous to the Mayari ore and therefore much interest attaches to the following comment of Professor Lacroix, p. 297:

"As for analysis (d) it reveals a complete disappearance of the silica. The interpretation of the constituents of the hydrates is uncertain. The red coloration, of variable shades, in the specimens studied leaves no doubt about variations in the hydration of the iron. If we assume that the hydrate of alumina is the one with three molecules of H_2O , the remainder of the water, which is combined in the hydroxide of iron, makes necessary for the latter a composition less hydrated than that of goethite. If on the contrary, we assume a hydrate of alumina with one molecule of water, then the hydroxide of iron would have the composition of goethite; but any such hypothesis ought to be rejected, since this mineral certainly does not exist in this laterite. Microscopic examination only proves the presence of a mixture of hematite and limonite."

⁴ Paul Krusch: *Primäre und sekundäre Erze unter besondere Berücksichtigung der "Gel" und der "schwermetallreichen" Erze* (Primary and Secondary Ores with Especial Consideration of the Colloid Ores and of the Ores Rich in the Heavy metals), *Compte Rendu de la XII Congrès Internationale Géologique*, pp. 275 to 286.

⁵ A recent paper by C. R. Fettke, *Limonite Deposits of Staten Island, N. Y.* (*School of Mines Quarterly*, vol. xxxiii, pp. 1 to 10, July, 1912), interprets the ores in accord with the most recent views.

⁶ Les Laterites de la Guinée, *Nouvelles Archives du Museum National d'Histoire Naturelle*, vol. v, pp. 255 to 356 (1913).

	Peridotite ^a	Peridotite ^b	Laterite ^c	Iron Capping ^d	Iron Ore ^e
SiO ₂	40.01	38.32	12.67		2.8
Al ₂ O ₃	2.54	2.66	12.59	4.80	8.7
Cr ₂ O ₃	0.16	0.16		0.20	tr.
Fe ₂ O ₃	1.00	4.35	46.84	83.50	77.2
FeO.....	11.70	11.78			
MgO.....	39.90	36.22	1.26		
CaO.....	1.68	2.74	0.04		
Na ₂ O.....	1.07	0.16			
K ₂ O.....	0.52	0.06			
TiO ₂		0.28	0.55		
H ₂ O combined.....	1.10	3.38	15.32	10.18	11.4
Insoluble.....			10.73 ^a	1.70 ^a	
	99.68	100.11	100.00	100.48	99.6

^aThe insoluble matter contains the chromic oxide in picotite.

In the Mayari ore, scales or needles which would correspond to the typical forms of goethite have not been observed, but as goethite is also believed to be at times massive or apparently amorphous the writer did not feel sure of its absence in the earthy and indefinitely flocculent mixtures to be seen in the ore when puddled with water under the microscope. Calculations based on this molecule therefore did not seem beyond the range of possibility. Professor Lacroix uses the special name of stilpnosiderite (an old term for a variety of limonite) for the colloid having the composition of limonite. Turgite, however, he concludes is a mixture of hematite and stilpnosiderite, such that the formula $2\text{Fe}_2\text{O}_3 \cdot \text{H}_2\text{O}$ is obtained. If a mixture corresponding to four hematite molecules and one limonite chanced to be analyzed, the formula of turgite would result. In French Guinea the same darkening of color is noted in the surface layer, and it is explained by Professor Lacroix as due to the dehydration of the limonite (stilpnosiderite) by the sun. The only two demonstrated hydrates of alumina in nature are believed by Professor Lacroix to be diaspore, $\text{Al}_2\text{O}_3 \cdot \text{H}_2\text{O}$, and gibbsite (or hydrargillite), $\text{Al}_2\text{O}_3 \cdot 3\text{H}_2\text{O}$. Both are known crystallized, but both appear to form colloids as well. Bauxite is not believed to be of definite composition, but to be a mixture of these two. So far as study of the Mayari ore has gone, no crystallized diaspore or gibbsite has been seen. We therefore appear to be dealing with a mixture of colloids and no inaccuracy of serious moment was involved in using the bauxite formula, $\text{Al}_2\text{O}_3 \cdot 2\text{SiO}_2$, which is practically a combination of one diaspore and one gibbsite. With superabundant combined water, an excess of gibbsite would be inferred, as in analysis II of the lean ore above. An extended literature has developed in recent years regarding the aluminum hydrates, but such new contribution to general knowl-

edge as it contains is chiefly based on the conceptions of colloids. The papers are, however, hardly germane for further citation here.

A very recently discovered body of ore, similar in all respects to the Mayari deposits, has been reported from the northern portion of the island of Mindanao in the Philippines. Its nature seems to have been first recognized by H. F. Cameron¹ of the Government Engineers' staff, who had had earlier experience in the Mayari district in Cuba. The ore is a surface mantle produced by the tropical weathering of basic, igneous rocks in latitude 9 N. Reserves of 800,000,000 tons are estimated and the following analysis is given:

SiO ₂	Al ₂ O ₃	Cr ₂ O ₃	Fe ₂ O ₃	NiO	H ₂ O	S	P	Total
1.20	12.20	1.28*	77.71 ^b	none	7.63	tr.	tr.	100.02

* 1.28 Cr₂O₃ = 0.88 Cr. ^b 77.7 Fe₂O₃ = 54.4 Fe.

This one sample is richer than the run of the Mayari ores by the year (48 to 49 Fe), but one would need to be assured by extensive boring and sampling that so high a yield as 54.4 could be maintained for long periods.

The thanks of the writer are due in full measure to Charles F. Rand, President of the Spanish-American Iron Co., for every facility to study the ores. Acknowledgments should also be made to the officers of the company at Felton and Woodfred for many courtesies.

¹ Personal letter to the writer.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Main Mineral Zone of the Santa Eulalia District, Chihuahua*

BY BASIL PRESCOTT, EL PASO, TEX.

(New York Meeting, February, 1915)

Résumé.—The district of Santa Eulalia lies 12 miles to the southeast of the city of Chihuahua, Mexico. The ore deposits occur in a Cretaceous limestone of unknown thickness, overlain by a series of rhyolitic tuffs and flows. The limestone is very gently folded and is cut by several systems of fissures, of which only those with a general north-south strike are pre-mineral and control the deposition. The deposits are of two main types; the earlier silver-bearing iron sulphide types occurs as high-grade tabular bodies resembling veins and as large masses of relatively low-grade ore. The gangue of this type is composed of lime-iron silicates that point toward a close association with an igneous rock mass which has not yet been encountered. In the later silver-lead-zinc type the ore occurs in vertical chimneys and horizontal mantas, principally as oxides, though at a depth of 1,500 ft. some sulphides have been encountered. During oxidation the ore chambers have increased in size owing to the solution of the walls by the sulphuric acid generated in the process, and a substitution has taken place of nearly one-half part by weight of the original constituents of the ore, by newly introduced elements and radicals. Both types are believed to have come from greater depths than those yet reached by mining, and their ultimate source, from a consideration of similar deposits, is believed to be an igneous rock in contact with the limestone at some horizon below. The deposits are believed to have been formed at unusually great depths, as indicated by the proved great vertical range of the silicate minerals coupled with the absence of the igneous rock, and by the practically unchanged character of the silver-lead-zinc ores as far as developed. The volcanic capping is believed to be distinctly later than the economically important ore deposition.

Introduction

THE Santa Eulalia district is situated in the State of Chihuahua, Mexico, and lies about 12 miles southeast of the capital, the city of Chihuahua. (See map, Fig. 1.) The district embraces the Sierra de La

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Santa Eulalia, an irregular eminence more or less circular in form, measuring from 4 to 6 miles in diameter at the elevation of the mines, with its

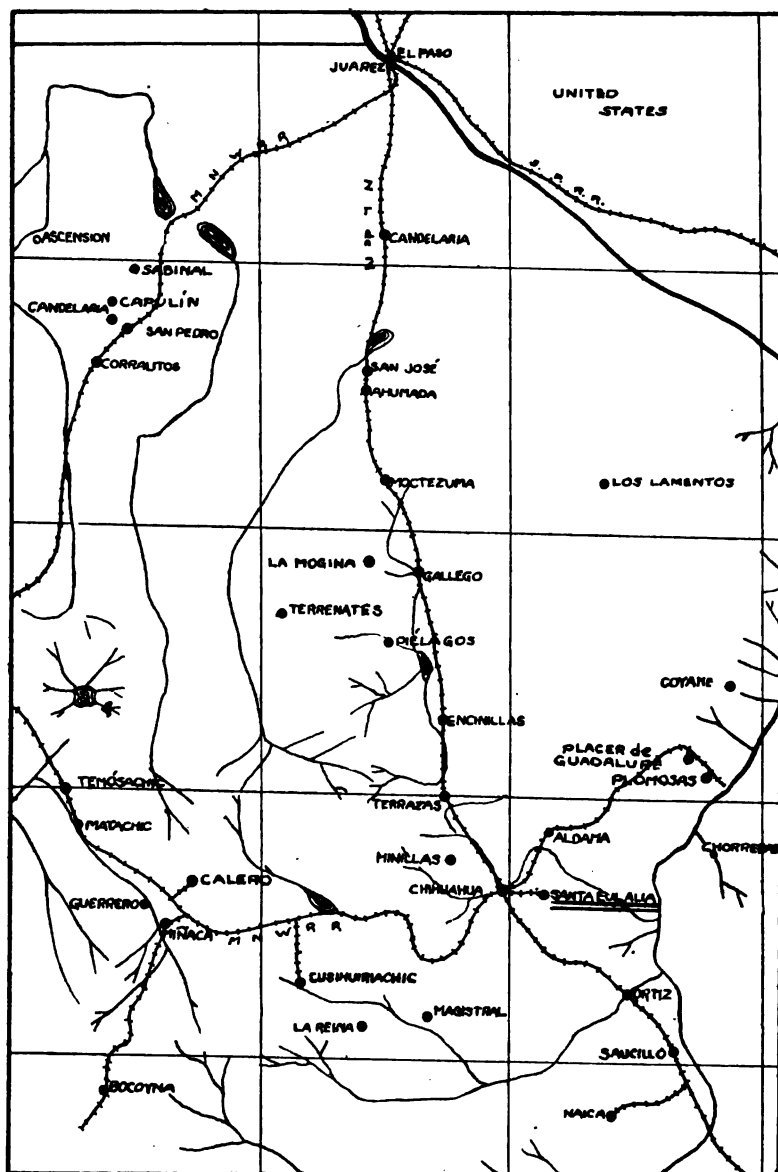


FIG. 1.—MAP OF A PORTION OF THE STATE OF CHIHUAHUA SHOWING LOCATION OF THE SANTA EULALIA AND OTHER MINING CAMPS.

base spreading over a much greater area, and rising to a height of over 7,000 ft. above sea level, or more than 2,000 ft. above the surrounding

plain. The sierra is part of the long, narrow Santa Eulalia range, but is surrounded by such deep, wide passes as to be practically isolated. At an earlier period it was a tableland, or mesa, which still shows in the topography and in the uniformity of elevation of the principal ridges and peaks; but the intense localized erosion typical of the arid climate of the northern plateau region of Mexico has carved wild, precipitous cañons and gorges into the very heart of the mountain. In one of its gentler, less picturesque cañons on the northwestern edge is located the Real de La Santa Eulalia, a typical little Mexican mining village.

The whole mountain is dry and barren, springs are practically unknown, and water is not always plentiful even in the town itself. With few exceptions all vegetation carries thorns or spines, and the only animal life that can survive the inhospitable conditions is composed of the flocks of goats and the scattered burros that in some strange manner exist from year to year.

The Real de La Santa Eulalia is reached from the city of Chihuahua by the Ferrocarril Mineral de Chihuahua, a narrow- (36-in.) gauge road that crosses the plain outside of the city, and, by heavy grades, follows the cañon up to the village. This road has daily passenger service, the trip out taking about $1\frac{1}{2}$ hr., and passes within a short distance of the reduction works of the American Smelting & Refining Co., to which plant it has a spur. The Chihuahua Mining Co. operates a 30-in. gauge road for the shipment of ores, principally those from its mine and those from the affiliated Potosi Mining Co., running from a point near the smelter on the Central Railroad up by the town of Santa Eulalia into the heart of the district, with spurs to several mines. This road is picturesque in the extreme, winding in and out of the cañons, with but little tangent once the town of Santa Eulalia is passed. The Central Railroad of Mexico, part of the National system, has a spur up to a point within a mile or two of Santa Eulalia, and an aerial tramway connects it with one of the mines several miles distant. A second aerial tramway runs out from the Mineral Railroad station at Santa Eulalia into the northern part of the camp.

The history of the Real de La Santa Eulalia and its mines is similar to that of many of the rich silver districts of Mexico, and has been fully and accurately given by many writers. Briefly it is as follows: According to some sources, the deposits were among the first discovered by the early Spanish prospectors, and 1591 is given as the date. However, official records of the State place the discovery at the beginning of the eighteenth century, in the year 1703, or about 12 years after the founding of the city of Chihuahua, a date that does not make the district an old one as Mexican mining districts go. The generally accepted story of its discovery is that it was made by a band of brigands, who, forced out of the mission town of Chihuahua, took refuge in one of the most inaccessible

parts of the Sierra de La Santa Eulalia, and from this point of vantage plied their trade. Such business could not then have been so good as in these last few years, or competition with the Apache Indians was too severe or too dangerous, for when silver and lead were discovered running out of their hearthstones, one night when they had an exceptionally hot fire, they immediately offered the information to the *padre* in Chihuahua in return for complete absolution. The inducement offered the *padre* was that he should be shown silver in such quantities as to allow him to build the greatest cathedral in all America, and while the cathedral of Chihuahua cannot claim that distinction, it certainly is a magnificent example of the architecture and construction of the early Spanish fathers.

The production of the district has been enormous, and though estimates vary widely it probably lies between \$300,000,000 and \$500,000,000. It seems reasonably certain that the total production for the first 86 years, from 1705 to 1791, amounted to approximately \$150,000,000, of which \$112,000,000 was reported and taxed.

The height of the bonanza days of the camp was during the last quarter of the eighteenth century, when the Real de La Santa Eulalia counted 6,000 souls, 63 *beneficios*, 168 reduction furnaces and 65 cupelling furnaces, while the city of Chihuahua had a population of 70,000, at least 20 smelting plants, and was practically supported by the activity of its mountain suburb. During the nineteenth century, owing to trouble with the Indians, the withdrawal of the Spaniards, and the political unrest of Mexico, the industry lapsed until the Americans entered the district in about the '80's. A most interesting and enlightening non-technical description of the conditions at Santa Eulalia at the close of the period of French occupation is given by General Lew Wallace in *Harper's Monthly* (November, 1867), while J. P. Kimball,¹ at about the same period describes the geology of the district, the occurrence of the ores, and the costs and methods of mining and reducing them. This paper will always be a classic in the literature of the district.

Following the entry of American mining engineers and American capital, came slowly improved facilities for transporting and reducing the ores; and yet more slowly, improved methods of mining and extracting them. It is a question just how far these improvements have succeeded in reducing costs, which is one of the ultimate objects of all installations of expensive equipment, but they have certainly increased the range of operations, making the deeper horizons as readily accessible as those near the surface; have increased enormously the tonnage that can be handled; and have reduced proportionately the losses during the reduction of the ores. High-grade and low-grade orebodies occur at all horizons, irrespective of the distance from the surface, and the Spanish

¹ *American Journal of Science*, 2d ser., vol. xlix, pp. 161 to 175

and Mexican miners encountered and worked both types. Some of the very oldest mines and deposits are now considered low grade, and, if judged by the ore since encountered in offshoots from the bodies they worked, and by the pillars and walls of ore that they left standing, even giving due allowance for their having extracted the richer portions, the limiting grade in the early days could not have been much above the limiting grade of to-day. This, I believe, is the surprising discovery that has often been made by those who have had occasion to examine old Spanish workings in various parts of Latin America.

Under American management and with American capital, the district has gone ahead until probably at no time in its history has the value in tonnage per year been so great as that of the last few years, and while a few of the mines may have passed their zenith, some of the largest producers look better to-day than ever before, and the district as a whole certainly has not reached its maximum annual production.

Numerous articles on the district have appeared in the technical journals,² while the district has been mentioned in connection with other Chihuahuan deposits in many of the papers on northern Mexico. M. A. Knapp³ gives a description of the eastern portion of the camp, an area not included in this paper; while the results of some of the most complete surveys and studies of the district have never been published. In the present paper, the writer has dealt only with the ore occurrences of the main mineral zone, for, while the rest of the district may be assumed to follow and be governed by the same principles and laws as those that appear to control the deposition within the area described, still it would require careful detailed study to ascertain that such was the case, and there is not sufficient development in the outside areas to warrant such work being undertaken at present.

The main mineral zone as treated in this paper includes the series of orebodies lying approximately N. 10° W., and limited on the south by the southern extension of the Potosi workings, and on the north by the northern limits of Mina Vieja, and including besides these two properties

² Argall, Philip: *Proceedings of the Colorado Scientific Society*, vol. vii, pp. 117 to 126 (1901-04).

Hill, R. T.: *Engineering and Mining Journal*, vol. lxxvi, No. 5, pp. 158 to 160 (Aug. 1, 1903).

Lakes, Arthur: *Mines and Minerals*, vol. xxiii, No. 12, pp. 529 to 531 (July, 1903).

Merrill, F. J. H.: *Mining and Scientific Press*, vol. xcvi, No. 1, pp. 37 to 39 (Jan. 2, 1909).

Rice, C. T.: *Engineering and Mining Journal*, vol. lxxxv, No. 25, pp. 1229 to 1233 (June 20, 1908).

Weed, W. H.: *Trans.*, xxxii, 396 to 399 (1901).

³ *Engineering and Mining Journal*, vol. lxxxi, No. 21, pp. 993 to 994 (May 26, 1906).

the Velardeña, the Santo Domingo, Buena Tierra, and San Toy mines.

General Geology

The oldest formation exposed in the district is a bluish gray limestone. (See Fig. 2.) This is probably Lower Cretaceous, belonging to the great Comanche series that once covered nearly all of Mexico; and while in the United States the formation is not generally of notable thickness, its importance rapidly increases to the southward and it is credited with over 4,000 ft. at the Rio Grande, and with the remarkable thickness of 20,000 ft. in central Mexico.⁴ It is the country rock of many mining

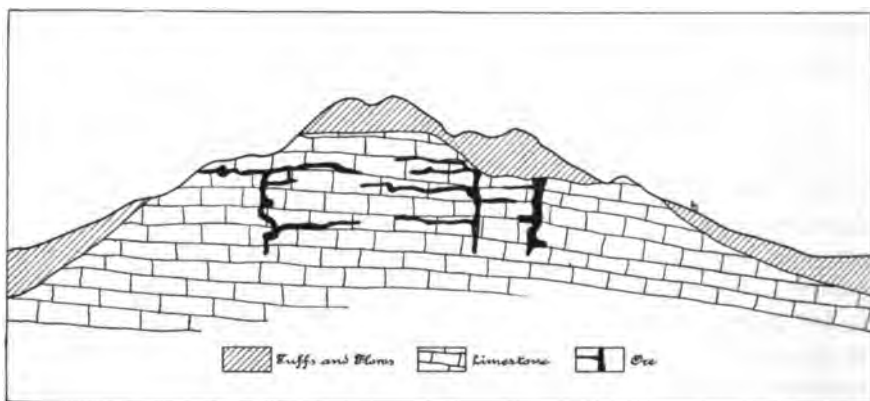


FIG. 2.—GENERALIZED LONGITUDINAL SECTION THROUGH THE ORE ZONE, SANTA EULALIA.

districts in the States of Zacatecas, San Luis Potosi, Coahuila, Nuevo Leon, Durango, Chihuahua and others, and has often a proved thickness as given by combined surface exposures and mine workings of from 2,500 to 3,000 ft., with ultimate depth and amount of erosion unknown. In many localities this limestone is conformably overlain by a shale series of great but unknown total thickness, but certainly often exceeding 3,000 ft., with a transition formation between, marked by several hundred feet of interbedded calcareous shales, and argillaceous limestones. In Santa Eulalia these shales are now lacking, and upon the eroded surface of the limestone is found a deposit of volcanic material and detrital matter, showing a maximum thickness of over 1,200 ft., apparently increasing in thickness as the Sierra Madre is approached toward the south and west, and thinning out near the valley of the Conchas toward the north and east.

The uplift and exposure of the limestone as a land surface took place at the close of the Cretaceous, and probably during the period of great

⁴ Hill, R. T.: *American Journal of Science*, 3d ser., vol. xlv, pp. 309 to 323 (1893).

igneous activity that ushered in the Tertiary. In Santa Eulalia this uplift and accompanying igneous activity caused little disturbance of the limestone beds, as dips exceeding 20° from the horizontal are practically unknown, and dips of from 5° to 10° are the rule over by far the greater part of the area. The resulting structure consists essentially of a series of gentle folds with approximate north-south axes pitching very slightly to the south, though at least one monoclinical east-west fold is known. Taken as a whole, moreover, these gentle dips yield a general anticline for the sierra, also with its axis north-south and pitching gently to the south. On the western edge of the sierra the dips are remarkably uniform to the southeast, these continuing back toward the northwest about $1\frac{1}{2}$ miles, at which point they reach the axis and lie practically horizontal or dip slightly to the southwest. This is the most important anticlinal axis of the sierra, as farther to the east the structure consists more of a series of minor folds and these are badly obscured by the capping of volcanic material mentioned above.

In character, the limestone is unusually uniform, there being no marked variation in color, texture, thickness of beds or chemical properties that will serve to distinguish one horizon from another. Fossils of a coarse crystalline calcite have been found at several horizons and, accentuated by differential weathering, have been noted over a large part of the surface of the district. Their distribution through the beds is somewhat irregular, and while variations are not surprising and do not interfere greatly with areal geology, they lead to some difficulties in plotting the conditions found in the relatively small exposures underground. Nevertheless, these fossil beds offer the only known method of correlation of the limestones throughout the camp. Two principal horizons are now recognized. The upper horizon occurs just below the capping in the higher parts of the area and is uniformly rich in fossils at this elevation, but the fossils extend downward irregularly, sometimes to a depth of 250 ft., usually, but not always, being less plentiful in the lower beds than in the uppermost 50 ft. The second horizon has been cut in the shafts and workings of various mines at a depth of about 1,000 ft. below the first, and has not yet been sufficiently opened up to determine its variations. While this horizon normally has a thickness of about 100 ft., it is practically lacking in some places and is nearly 200 ft. thick in others. In many places throughout the limestone, both in the fossiliferous beds and in the intervening non-fossiliferous stratum, chert nodules appear, arranged usually parallel to the bedding, but not sufficiently confined to definite horizons nor sufficiently continuous to be of any value as a factor in correlation.

In general the limestone is compact, semicrystalline, varies in color from light gray to dark bluish gray, and chemically is nearly pure calcium

carbonate. The fossil beds carry sufficient hydrogen sulphide to give a fetid odor.

The uplift of this limestone was followed by a long period of erosion, for not only was every vestige of the overlying shales removed, but even the intermediate series of interbedded shales and limestone are lacking in the Santa Eulalia district. The topographic features at the opening of the Tertiary and the laying down of the first of the volcanics consisted essentially of the same sharp sierras as to-day, but with the flat-topped tableland more prominent and with the cañons radiating from it less precipitous and more sloping than at present. A good idea of the old topography may be drawn from those parts of the district where the capping of tuff has been eroded away and the limestone has been again exposed, this time to the action of the alternate long dry spells and torrential rains. The drainage was essentially then, as now, in all directions from the central portion of the sierra or tableland; but the exact locations of the *arroyos* of to-day are not those of the old surface, nor has the present erosion in the southern and lower areas reached the bottoms of the old cañons, which is perhaps due to the raising of the level of the valley by the addition of vast deposits of detrital matter.

The overlying capping is made up of a series of tuffs and flows, probably from the Sierra Madre to the west, and accumulations of detrital matter from some higher area, probably lying to the north, and since removed by the Conchas river and its tributaries. The series is interesting, but economically and geologically unimportant, and varies greatly in different parts of the sierra. As the district was high land at that time these accumulations were laid down as continental deposits and are not so distinctly stratified as though laid down under water. Nevertheless, they show distinct bedding and occasionally some sorting.

Upon the eroded surface of the limestone, where the topography of that time would permit, an accumulation of a few meters of conglomerate is often found. Above this lies the first tuff, usually rhyolitic, and of a thickness depending upon its location with reference to the old topography, filling first the hollows and cañons between the steep hillsides and tending to make the surface of the land more even, a tendency which, at the same time, was offset by the continual process of erosion. Following this is a flow of rhyolite, again of very variable thickness and again exposing an eroded but more even surface to the next of the series, a mixture of rhyolite and andesite tuff. This deposition of rhyolite and andesite tuffs and flows, interrupted by erosion periods, continued until the total thickness must have been considerable and the old topography was obscured.

At the present time this capping as a whole is not a very resistant rock, certainly not to be compared with the limestone below, and as a result has been rapidly though smoothly eroded. The main flows of rhyolite

and the tuffs, when they have been silicified by later solutions, offer exceptions to the general rule and form topographic features second in ruggedness only to those of the limestone itself.

General Economic Geology

Three distinct types and ages of ore, controlled by fissures, occur in irregular deposits in the limestone. Of these the earliest was a silver-bearing iron sulphide with a mixed lime-iron-alumina silicate gangue, that has been exposed for some years but has only recently become of economic value by the discovery of certain large portions running high in silver. It occurs in enormous masses, much of it too low grade for mining at the present time, but some of it highly profitable. Extending vertically from these ore masses are tabular veinlike feeders, often 100 to 200 m. long by 10 to 15 m. in width.

The second type is the one for which the district is famous. It has been mined for the last two centuries and belongs to the silver-lead-zinc ores common to replacement deposits in limestone. The ore occurs in chimneys and "mantas" (beds or blankets). The chimneys are more or less circular or elliptical in form, often slightly extended along the fissures, though showing all the variations and irregularities common to deposits in limestone. They extend from the greatest depths reached, up to various horizons, and sometimes stand nearly vertical for 1,000 ft. or more. The mantas make off from these chimneys along favorable horizons, lie practically horizontal, and follow roughly the pre-mineral fissures for great distances, often over 1,000 m., to a maximum of perhaps two or three times that distance. The chimneys measure from 20 m. in diameter up to 100 m. in width and 200 m. in length; the mantas from 20 m. or less in width by 10 m. or less in thickness, up to 100 m. in width by 50 m. in thickness, occurring locally even larger, and all variations and combinations being found. With one or two minor exceptions, where square setting was tried by some of the earlier Americans in the district, these bodies have been mined without timber, and as the limestone usually stands well, most of these enormous old stopes are still accessible.

The third type has no economic importance, as apparently workable deposits do not result from it. The ore is a pure galena without admixtures of zinc or iron sulphides, carrying a little silver and practically no gangue, and its deposition took place after the formation of all fissures, and hence long after that of the primary ores of economic importance. It is apparently the result of an expiring phase of mineralization. The occurrence is typically a narrow streak of galena, usually an inch in width, deposited along the fissures of all classes, but best developed in the latest. Local enlargements resulting in small chambers or mantas have not yet been found and are probably entirely lacking.

Both of the economically important types occur as oxides or carbonates, and as sulphides, the sulphide ores greatly predominating in the silver-bearing iron sulphide type; while in the silver-lead-zinc type the oxidized or carbonate ores have been in the past, and are still, the principal producers. Until comparatively recent years the silver-lead-zinc ore was the source of the entire production of the district; but at the present time a small but increasing tonnage of the silver-bearing iron sulphide ores is being produced, perhaps representing one-fifth of the total production of the camp.

Before describing in more detail the two economic types of ore occurrences, their probable genesis and the laws that seem to govern their deposition, a classification of the fissures and dikes and a list of the ore minerals will be given, as these form the basis for many of the statements made.

Fissure Systems

In the limestone, and in certain cases in the overlying igneous capping, are developed certain definite sets of fissures and fault planes. When compared with the fissures found in other mineral districts, these are remarkable for the uniformity of those characteristics which serve to distinguish the fissures of one set from those of another, and, when considered broadly, for their continuity and persistence. This is doubtless due to the uniform character of the limestone, the absence of intense folding, and the position of the fissures—standing nearly vertical or practically normal to the limestone beds. All the various fissures have the characteristic usually shown by those found in limestone of closing and opening with bewildering rapidity, both along dip and strike, though this does not apply equally to all groups.

Classification of Fissures

	Group	Class	Economic Characteristics
Earlier than primary mineralization of economic importance.	1	N-S.	Principal producers of silver-bearing iron sulphide. Large producers of silver, lead, zinc.
		N. 10° E.	Variation of N-S. class.
		N. 10° W.	Principal producers of silver-lead-zinc ore.
		N. 30° W.	Variation of N. 10° W. class.
Later than primary mineralization of economic importance.	2	N. 55° E.	Principal fault planes and channels followed by dikes.
	3	N. 70° W. to E-W.	Principal producers of secondary zinc ore, to date.
	4	N. 30° E.	Principal channels of oxidation.

In the accompanying table the fissures are grouped according to their strike and arranged progressively according to age.

Group 1.—Probably the earliest and the only certain pre-mineral fissures that in no case are later than the ore deposition are those of the North-South group. The direction corresponds with and probably results from the North-South fold, and the fissures are more commonly found along the broad axis of the anticline than elsewhere, though known to occur frequently on either limb. They mark the general direction of nearly all orebodies of the district; no primary ore of economic importance has yet been found that does not occur along one of these fissures. They pass through all chimneys and serve to determine the course of all true mantas over the major part of their distance. None of them has been found passing out of the limestone and into the overlying tuff. They stand nearly vertical, seldom varying 10° from it, even locally, though sometimes, on the limbs of the anticline, there is a tendency for the fissures to stand perpendicular to the beds of limestone rather than vertical. In general the group is remarkable, aside from its productivity, for its tightness and inconspicuousness in the country rock.

This group may be subdivided into four classes. The first or North-South class includes such well-known fissures as the Hematite and Velardeña fissures and those governing the San Toy manta. The second or N. 10° E. class appears usually as a branch or offshoot of the North-South class and is not very continuous either along dip or strike. The third or N. 10° W. class includes such fissures as the Purisima, Los Angeles, Santo Domingo, and Potosi, while the fourth class, or N. 30° W. fissures, is apparently a variation of the N. 10° W. class, caused by a slight irregularity of the anticlinal axis that occurs in the area to which they are confined.

The North-South and N. 10° W. have not been found typically intersecting one another and, from a consideration of the fissures alone, it might seem that they were simultaneously formed, but, as will be shown later, it is probable that the true North-South fissures were earlier. In some cases the different classes occur in the same part of the district, as for example the Hematite and Potosi fissures; but in general each is more or less restricted to predominance over certain areas. For example, very few North-South fissures are found in the northern half of the developed zone, the N. 10° W. fissures being the rule up to the extreme northern limits, where the N. 30° W. class appear almost to the exclusion of other fissures of the North-South group.

It is not an easy matter to determine the continuity of any one of these fissures, both the N. 10° W. and North-South classes affording examples of single fissures that appear to extend over long distances, such as 500 m. or even more, but as a rule the great extent is arrived at by assuming that developed segments will connect continuously. In the upper horizons

many of the fissures represent a series of parallel fractures, no one of which is continuous for more than 50 m., but which as a group and under a single name may be continued for a kilometer or more.

In one or two cases slight fault movements of several meters are known to have taken place along the N. 10° W. fissures after the deposition of the ore, but these cases are relatively rare, and in general none of this group can be said to belong to the type of fault fissures, any noticeable movement being decidedly later than the period at which they were first formed, and representing simply a more or less accidental adjustment of stress along a previously existing plane. The Los Angeles fissure, for example, shows a marked amount of attrition and is accompanied in certain places by heavy gouge and well-developed slickensides. In the central portion of the developed zone, certain N. 10° W. fissures carry a small and variable amount of dike material, apparently representing offshoots from the main dikes, as will be explained later.

The exceptions to the above-noted directions of North-South, N. 10° W., N. 10° E., and N. 30° W. are not numerous; in fact, probably not one case in a hundred will vary from them in the main zone developed by the principal mines. As a rule, when measured over short tangents, the fissures lie within 5° of the given directions, and when the strike is taken over extended distances the variation is even less. It seems probable, however, that in other areas different conditions will prevail, resulting in a slightly changed strike for the whole system, in the same way that the North-West fissures appear to have resulted locally from the N. 10° W. class.

Group 2.—The N. 55° E. fissures form the second group. As a whole this group is characterized by a truly remarkable regularity in strike, for it is nearly impossible to note a variation with a hand compass, and it shows the same uniformity of strike when plotted over long distances. The fissures may vary more in dip than do the North-South group, several of the more important examples dipping from 15° to 20° from the vertical. As a rule they are open for a greater part of their extent than those of the North-South group, and are often filled, especially below an orebody, with products of alteration and secondary deposition. They are the most prominent fault planes of the district, two cases in particular being noted where the movement has probably exceeded 20 ft. and in one case may even reach 50 ft., and innumerable cases are found where the movement has been from a few inches to 10 ft. Most of the dikes follow these fissures with offshoots from them into the other fissures.

The movements, and apparently the fissures themselves, are later than the primary ore deposition of economic importance, earlier than the oxidation and secondary deposition, earlier than the deposition of the capping, and much earlier than the dikes. Some of the movement, however, was later than the capping and a few cases have been noted where these

fault fissures cut through the lower members of a capping series. The more important faults are accompanied by a coarsely crushed zone several feet wide. Where slightly inclined, the faults, as far as have been observed, are normal, and in the central part of the area are more or less compensating, as the fossiliferous horizons show. This N. 55° E. fault fissuring is exceedingly common, an example occurring every few meters, and while the majority are but knife-blade cracks, well-defined fissures are prevalent.

The exceptions are rare, and it is difficult to determine whether or not they should be treated with this group. A strong N. 45° to 50° E. fissure, called the San Lazaro, cuts the Santo Domingo chimney or *chorro*, which with several parallel associates has some of the characteristics of the pre-mineral fissures. These characteristics, however, are found only in the vicinity of the mineralized North-South group, and it is probable that the fissure itself is not pre-mineral. Although there are no evidences of later movement along these fissures, they appear to cut the capping and possibly should be included under the head of the N. 30° E. fissures to be described later.

Group 3.—The N. 70° W. to East-West fissures form the third group in order of time. These also are fault planes, not usually so marked as the N. 55° E., and probably much more recent. They are not common, although, where found, they are well developed, a few having been traced over distances of several hundred meters and to a depth of 1,000 to 1,500 ft. They are not very constant either in direction or dip, but do not vary far from the vertical. As a rule they represent one of two extremes, either so tight as to attract no attention, the usual case; or so open as to form one of the principal channels for the secondary solutions and for ordinary circulating waters. Although the movements along them have not been determined, they do not appear to have been great, certainly never more than a few meters; but they are accompanied in well-developed cases by a zone of coarsely crushed limestone, which sometimes shows a thickness of from 10 to 15 m. The Gypsum fissure is the best known example of this type. As in the case of the N. 55° E. system, these fissures in rare instances cut the tuffs; but many of the important examples do not do so, and it seems probable that their formation and most of the movement antedate the capping. They are also earlier than the dikes and oxidation and later than the primary mineralization of economic importance and the first two groups of fissures.

Group 4.—The fissures of the N. 30° E. system compose the fourth group. They are later than all other fissures and later than the capping, which they cut, outcropping with remarkable distinctness on the surface of the tuff and porphyry, sometimes for a distance of several hundred meters. They are also later than the dikes, the only dike material found in them being near their intersection with the dike-bearing fissures, and

this representing material, largely clay, washed into them mechanically, as opposed to the occurrence in other fissures where it has been intruded into the open spaces in a molten state. They cut and offset the N. 55° E. fissures, which in turn cut and offset the orebodies of the North-South group. They are fault fissures of very minor movement, seldom showing any crushing of the walls. They are the most pronounced, most numerous and best developed fissures in the district, and in the past, before their barrenness and age had been determined, a great deal of non-productive exploration was done along them. They are of slight economic importance—only as they form channels for oxidizing solutions, and as such may serve as a guide to development. In only one place in the main mineral zone is their relation to other fissures and the ore obscure, and that is in the area before described, where the San Lazaro and associated N. 45° to 50° E. fissures appear in doubtful relation to the orebody. In this area, the trend of the primary orebodies, now oxidized, is approximately N. 30° to 40° E., apparently following one or both of these two sets of fissures out of the Santo Domingo chimney; but irregularities in the mantas in the vicinity of the main chimneys are the usual occurrence. Judging these fissures by the remainder of the district and the manner in which they intersect other groups, it seems probable that the ore deposition was controlled by some other factor than the course of these fissures.

The Dikes

Two classes of intrusives are found in the zone, andesitic and rhyolitics. They do not occur very abundantly nor are they well developed. Unfortunately, also, they have not as yet been found intersecting, hence their relative ages have not been determined. Their relation to the capping does not afford the desired information, as they both cut the capping series and outcrop on the surface. The relation of both to the ore deposition is the same; that is, decidedly later than all primary deposition of economic importance, later than all fissures except those of the N. 30° E. group, and earlier than the oxidation and alteration of the orebody. Both the andesite and rhyolite dikes followed the pre-existing N. 55° E. fissures as main channels, but at their intersection with the North-South group and the N. 70° W. to East-West group they are often found following these fissures for distances up to 100 m. In fact, they show conclusively which fissures, water courses, caves, etc., were open at the time of their intrusion, as they apparently filled every available open space.

The andesite dikes are comparatively fresh and unaltered on the surface and in those parts of the mines far removed from orebodies, but in the vicinity of the oxidized ores are almost completely altered to kaolin, probably by the action of sulphuric acid solutions. They occur both at the northern and southern extremities of the developed area, in the latter,

the Potosi mine, reaching their maximum development in a single dike nearly 20 ft. wide.

The rhyolite dikes occur throughout the developed area and are characteristically smaller, seldom exceeding a meter in width, and are even more intensely altered than the andesite dikes. A fairly fresh sample from the Potosi mine gave an analysis closely approximating that of normal unaltered rhyolite.

Ore and Gangue Minerals

In Table I the ore and gangue minerals are listed in four groups, the first three corresponding to the three types of ore, and the fourth, in limestone, including those minerals developed in the channels of the limestone, often so far removed from their source that it is difficult to refer them definitely to any type. Alunogen, occurring as an efflorescence on limestone walls, is an example, for the sulphate radical may be derived from the oxidation of any type of ore, and the alumina from either the minute quantities contained in the limestone, from the silicate gangue of the silver-bearing iron sulphide type of ore, or from the alteration of the dikes. The subdivisions, primary and secondary, distinguish the minerals of the sulphide ore from those of the oxide or carbonate ore—that is, those formed by ascending solutions from those formed by descending solutions—with the one exception that the quartz, which was introduced after considerable of the oxidation was complete, probably came from below.

The list is certainly not complete, since the purpose of the examination was not such as to warrant the investment of time in the search for and determination of mineral species. However, the list probably contains a sufficient number of critical minerals to classify the deposits according to our present ideas of economic geology.

These minerals are all common and well known, excepting perhaps ilvaite, fayalite, and kenebelite, and it is probable that the field would prove an attractive one to a mineralogist in search of the rarer minerals.

The Silver-Bearing Iron Sulphide Type Ore

There are very few known occurrences of the silver-bearing iron sulphide ore in the Santa Eulalia district, and these have all been encountered within the last 10 or 15 years. Several factors have, unfortunately, retarded the development of the type: First, the ore where first encountered was largely of a grade that at that time offered little inducement to further development; second, the occurrence was considered as a local phase of the silver-lead-zinc type, and as such was of little importance; and third, where encountered in the early stages of development, it was often intermingled with the later type of lead-silver-zinc ore, and its true nature

TABLE I.—*Occurrence of Ore and Gangue Minerals*

	Silver-bearing Iron Sulphide Type		Silver-lead- zinc Type		Pure Galena Type		In the Lime- stone
Pyrrhotite.....	Prim.		Prim.				
Pyrite.....	Prim.	Sec.	Prim.	Sec.			
Marcasite.....							Sec.
Arsenopyrite.....	Prim.						
Magnetite.....	Prim.						
Hematite.....		Sec.		Sec.			
Turgite.....		Sec.		Sec.			
Limonite.....				Sec.			
Copiapite.....							Sec.
Melanterite.....							Sec.
Siderite.....			Prim.?	Sec.			
Manganite.....			Prim.?	Sec.			
Psilomelane.....			Prim.?	Sec.			
Wad.....				Sec.			
Pyrolusite.....				Sec.			Sec.
Rhodocrosite.....				Sec.			Sec.
Rhodonite.....				Sec.			Sec.
Sphalerite.....	Prim.		Prim.	Sec.?			
Smithsonite.....				Sec.			Sec.
Calamine.....				Sec.			
Hydrozincite.....				Sec.			
Goslarite.....							Sec.
Willemite.....				Sec.			
Greenockite.....				Sec.			
Galena.....	Prim.	Sec.	Prim.	Sec.	Prim.		
Anglesite.....				Sec.			
Pyromorphite.....				Sec.			
Mimetite.....				Sec.			
Cerussite.....				Sec.			
Wulfenite.....				Sec.			
Knebelite.....	Prim.						
Chalcopyrite.....	Prim.		Prim.?				
Chalcocite.....				Sec.			
Brochantite.....				Sec.			
Malachite.....				Sec.			
Azurite.....				Sec.			
Native silver.....				Sec.			
Cerargyrite.....				Sec.			
Argentite.....	Prim.		Prim.				
Proustite.....	Prim.						
Gold.....	Prim.						
Mirabilite.....							Sec.
Epsomite.....							Sec.

TABLE I.—*Occurrence of Ore and Gangue Minerals.—Continued*

	Silver-bearing Iron Sulphide Type		Silver-lead- zinc Type		Pure Galena Type		In the Lime- stone
Alunogen.....							Sec.
Halite.....							Sec.
Barite.....			Prim.				
Fluorite.....	Prim.		Prim.	Sec.			
Calcite.....	Prim.	Sec.	Prim.	Sec.			Sec.
Aragonite.....							Sec.
Dolomite.....				Sec.			Sec.
Gypsum.....		Sec.		Sec.			Sec.
Sulphur.....				Sec.			
Quartz.....	Prim.	Sec.	Prim.	Sec.			
Chalcedony.....	Prim.						
Ilvaite.....	Prim.						
Hedenbergite.....	Prim.						
Chlorite.....	Prim.	Sec.					
Fayalite.....	Prim.						
Uralite.....		Sec.					
Tremolite.....							Sec.

thus obscured. Several years ago, however, the discovery was made that great tonnages of this type carried high silver percentages, making such ore extremely desirable, and recently a clearer conception of the occurrence and characteristics of the type was obtained, with the result that in the future it should be an increasingly important factor in the production of the district.

The following partial analysis of a sample of the ore, made by W. E. Soest for this purpose, gives a good conception of the character of this type, even though the sample itself may not be the average occurrence:

Pb	Zn	SiO ₂	CaO	BaSO ₄	Fe	Mn	S	As	MgO	Al ₂ O ₃	Total
0.2	0.5	11.2	4.6	1.4	39	5.1	14.6	2.5	1.2	3.6	83.9

Recalculating this analysis on the basis of its known mineralogical content, gives: Galena, 0.23; sphalerite, 0.74; arsenopyrite, 5.43; pyrite, 3.75; pyrrhotite, 26.77; barite, 1.4 per cent.; the remainder, consisting of Fe, 19.69; CaO, 4.6; Mn, 5.1; MgO, 1.2; SiO₂, 11.2; Al₂O₃, 3.6 per cent., cannot be readily recalculated.

The conditions in Mexico during the last year have been such that a microscopic examination of this ore has not been made, as planned, but an examination of a number of specimens shows the presence of several rare silicates. An analysis of one of these by W. E. Soest gives: SiO₂,

28.9; Fe, 38.9; CaO, 10; Mn, 4.2; and H₂O, 2 per cent., which checks the previous determination of ilvaite. Under the microscope this mineral is opaque in the thinnest section obtainable, but fine-crushed fragments show the typical pleochroism of ilvaite. Magnetite occurs throughout the sections examined so finely divided that it was not previously recognized in hand specimens, and associated with it is the ferrous manganese silicate knebelite and the ferrous silicate fayalite. As the latter has not been reported in ore deposits, great care was taken in its determination, and a number of thin sections were examined in which fayalite was the most prominent mineral. Its index of refraction, determined in fragments immersed in various liquids, is greater than 1.83 and less than 1.93, and its birefringence, tested with a quartz wedge in sections showing some quartz, is about 0.05. An analysis by William J. Van Sicklen shows SiO₂, 28.3; FeO, 55.6; MnO, 15.3; CaO, 0.0; total, 99.2 per cent. Another analysis, by W. E. Soest, shows SiO₂, 29.1; FeO, 49.1; MnO, 19.1; CaO, 0.1; total, 97.4 per cent. Other tests show even greater variation in the iron and manganese, due, probably, to isomorphic mixtures of knebelite and fayalite. In some cases it has been subjected to hydro-thermal alterations resulting in a brownish mineral somewhat resembling iddingsite, which could not be determined, chlorite, chalcedony, and magnetite. The chlorite, however, usually occurs as a direct replacement of the limestone, and although the magnetite is invariably associated with the fayalite and knebelite, the latter are seldom altered. These minerals are not present as rare constituents but as a group normally form nearly half of the ore, each locally predominating, and each probably aggregating thousands of tons.

From a consideration of the mineralogical content and chemical analysis, the occurrence might well be included under the head of contact-metamorphic ore deposits, presumably with limestone for one wall and an igneous rock for the second; and from the low amounts of alumina, the great quantities of the basic lime-iron silicate, ilvaite, the absence of copper sulphides, and the presence of argentiferous pyrite and arsenopyrite, it might further be classified as belonging to the last stage of contact metamorphism. On the other hand, the presence of the large quantities of magnetite and fayalite would seem to point to the intense physical conditions found only in a magma or in close association with it. Although the specimens examined microscopically came from a depth of about 500 ft. below the contact between the tuff and limestone and at least 600 ft. above the bottom of the mine, no igneous rock has yet been found in the main zone aside from the few narrow dikes unaccompanied by metamorphic action and the volcanic detritus of the capping.⁵

⁵ In the eastern camp, there is a stronger dike with silver-lead-zinc ore along its contact but unaccompanied by contact-metamorphic minerals.

In shape, form, and dimensions, the type shows considerable variation, but the economically important occurrence is a tabular, veinlike body, 5 to 15 m. in width, standing vertical and developed downward 500 ft. below the capping, with ultimate limits in depth unknown. The walls are parallel and show as little variation and irregularity over great distances as do the walls of the average vein. The ore is usually frozen to the limestone, but there is no intergrading of the ore with the wall rock and in the economically important occurrences no noted development of silicate minerals in the limestone.

The ore itself is usually banded vertically, parallel to the walls, apparently not developed as incrustations in an open fissure, but as mineralization along parallel fractures of the same fissure with later replacement of the included country rock; and where this replacement has not been entirely completed and the white limestone horses, a foot or more in width, are included in the dark ore, the banded appearance of the orebody is very strongly emphasized.

All the ore of this type found so far occurs along the true North-South class of fissures and there is reason to believe that mineralization of the silver-bearing iron sulphide type is confined to them to the exclusion of the other classes of the North-South group; although definite proof is not yet obtainable, it seems probable that this mineralization usually antedates the formation of most of the other fissures which, when considered from the standpoint of the silver-lead-zinc type, are pre-mineral. The characteristic occurrences of economic importance extend along these North-South fissures for 100 to 150 m. and sometimes much more, and then thin out with remarkable suddenness, an orebody of nearly average width being sometimes reduced to a scarcely perceptible fissure within a couple of meters. Along the same fissure, however, several orebodies may occur, widely separated by stretches of barren limestone in which the fissure itself may be practically imperceptible and indistinguishable from the numerous, small, unmineralized breaks of the limestone.

Bodies of oxidized ores of this type have been found, but these are relatively rare, as the character of the mineralization and the density and texture of the ores are not favorable to rapid oxidation, and the type is considered from the standpoint of sulphide only. The known oxides were originally low in gangue and high in metallic sulphides, so that the resulting ore is predominately an oxide of iron, usually hematite; but a few minor occurrences of the semi-oxidized silicates show that the pyroxene usually breaks down into amphibole, a soft brownish uralite, while the alteration products of the more resistant ilvaite have not yet been isolated.

The principal variations from the above type occurrences are found usually where very favorable limestone horizons are crossed by the

ore channel. Working down on one orebody, it was found to widen rapidly from the normal occurrence until, within something over 100 ft. of additional depth, it was nearly as wide as long, yielding a great tonnage of excellent ore. The most interesting and complex occurrence of this type of mineralization is an enormous mass not yet completely developed, but showing at present 1,000,000 tons or more of mineral matter. Portions of this mass are high grade; the remainder is simply a limestone, replaced by silicates, carrying a small proportion of metallic minerals. The feeders of this orebody are the North-South fissures, several in this case, marked by the vertical banding of their contents, but this banding is found standing at increasingly flat angles as the fissures are left, until the greater part of the area is a horizontally banded, completely metamorphosed mass with the width nearly equal to the length, and a depth, outside of the feeding fissures, much less than either length or width. This mass connects continuously with one of the high-grade tabular bodies and has itself high-grade portions, and furthermore in places has been enriched by the addition of impregnations of the silver-lead-zinc type from a chimney of that ore which touches this body on one edge. This admixture was the ore first encountered, and it is not remarkable that the occurrence should have been considered as merely a local phase of the silver-lead-zinc mineralization, as it appeared to grade into a typical body of that type. A careful examination shows, however, that this silver-lead-zinc mineralization was introduced later and was distinct from the main development of metamorphic minerals and deposition of iron sulphide, for, particularly where the enrichment has not progressed far, it may be noted that it consists largely of a filling of open spaces in the pre-existing ore. The body, since its formation, has been cut by all the post-mineral fissures of the various classes, and these stand vertically and have slightly shattered the ore for a few inches in width, rather than making simple fissures as in the limestone; and these shattered zones have been filled with calcite and secondary quartz, rhodochrosite, pyrite, sphalerite, and galena, perhaps in the order named, this mineralization being secondary and later than the enrichment above described, for these fissures cut through the portions of the orebody enriched by the lead-silver-zinc type as clearly as they do through the unenriched metamorphic ore. These fissures are locally very plentiful and give to the deposit a false impression of vertical banding, similar to that found in the North-South feeders, particularly as some of the more soluble lime-iron silicates have been rearranged parallel to and along the edges of the fissures. Continued examination shows that the silver-bearing pyrite type, locally high in gangue and low in metallic content, lies horizontally bedded at all points excepting along the North-South feeder fissures, and that the cross fissures cut this banding at right angles.

In summation, it is believed that this type of ore is the earliest miner-

alization yet exposed in Santa Eulalia; that a time interval elapsed between its deposition and that of the silver-lead-zinc type; that all cross fissures were later than the mineralization; also that the ore rose along the fissures practically as veins, though not continuously, there being intervening barren sections. On certain favorable horizons the mineralizer made off into the surrounding rock, metamorphosing it into a low-grade body of silicate ore which in some cases has been enriched by additions of the later silver-lead-zinc type.

It is important that this type of ore be distinguished readily from, for instance, the low-lead class (see below) of the later silver-lead-zinc type, for the differences in the line of development pursued are so great that the development of one does not accomplish the development of the other. This distinction may be readily made even on a small exposure, by the presence of arsenopyrite in detectable quantities in the metallic content of the earlier type; the presence of dark silicates or their alteration products in its gangue; by its prevalent vertical banding in the pay ore as opposed to the almost universal horizontal banding of the lead-silver-zinc type, and, if sufficiently developed, by its tabular form.

The Silver-Lead-Zinc Type Sulphide Ores

Clean primary sulphide ores of the silver-lead-zinc type are nearly lacking in the district up to the depths yet reached. Several chimneys show undeveloped sulphide ores in their lowest horizons and one of these occurrences has been developed for several hundred feet, but this body has been partly inaccessible for some time, and has oxidized with surprising rapidity since it was first exposed. There is another occurrence of sulphide ore in the district which has every indication of being primary, and unenriched by secondary solutions carrying metals, but this, unfortunately, had not been completely developed at the time of this examination, so that its general nature, extent, and relation to other nearby bodies of carbonate ore could not be determined. Strangely enough, this occurrence, instead of being in the deepest explored horizons, is found but a short distance below the capping, in places separated from the volcanic material only by a comparatively minor occurrence of oxidized ores. It has been noted, however, that the occurrence of sulphides depends almost entirely upon the local channels of circulation of the oxidizing waters, rather than upon the depth below the surface, and one of the factors governing these waters is the thickness and condition of the more or less impervious capping. It has been further noted repeatedly that near an open channel, where the waters have had free passage, instead of being completely oxidized, the ore may be partly altered, the supposition being that the unretarded waters did not then percolate through the whole mass as they did when the channels were more devious and

less well defined. In this case, it seems probable both that a nearby N. 70° W. fissure offers the open channel that drains the oxidizing solutions traveling along the contact away from the area, and that the unfissured condition of the impervious capping at this point protected the body from above.

Some difficulty is encountered when the reconstruction of an average normal primary sulphide ore is attempted. The commercial analyses of shipments to the smelters and the results of systematic sampling are the only records covering a sufficiently great quantity of ore to give an idea of its average content, but these analyses are run only for commercially important elements and oxides, and the samples often include wall rock, usually include portions somewhat oxidized, and often portions high in secondary sulphides. The occurrences of sulphide ores are not numerous nor in all cases accessible, and their variations are considerable. It seems possible that certain of these variations are due to the depth at which the ore is found, so that even if a clean primary ore were encountered at a deep horizon, it is doubtful if this would represent the original ore of the oxidized bodies found above.

The following are partial analyses of the sulphide ores, No. 1 representing a composite of many samples and the work of many assayers, Nos. 2, 3, and 4, analyses made by W. E. Soest for this work, and No. 5 a selected composite of several samples, analyst unknown.

No.	Pb Per Cent.	Zn Per Cent.	Fe Per Cent.	SiO ₂ Per Cent.	Mn Per Cent.	CaO Per Cent.	S Per Cent.	Total
1	15.	14.	28.	4.	0.8	2.6	24.	88.4
2 ^a	13.3	12.	22.7	4.	2.1	10.7	20.5	87.6
3	2.	4.	17.	16.	1.5	10.	25.	75.5
4 ^b	26.6	12.5	30.	0.4	0.5	0.4	28.2	99.2
5	12.7	10.3	36.	2.2	1.	1.5	28.1	91.8

^a Also: As, 1.3; Mg, 0.8; BaSO₄, 0.2

^b Also: As, 0.2; Al₂O₃, 0.4.

No. 1 is an average analysis of one of the few large sulphide bodies now exposed in the district, but when it is considered in detail it is immediately evident that the ore is far from being primary and unaltered. It is useless to attempt to recalculate the analysis, as the sulphur is insufficient to satisfy the iron, lead, and zinc, as sulphides, of any known mineral species; but the ore is believed to be composed of galena, sphalerite, and pyrrhotite—the latter to the practical exclusion of pyrite—sulphates of the same metals, a trace of arsenopyrite, and a gangue of quartz and mixed carbonates with a little oxide of iron. This would satisfy the

analysis, but the result is an ore on which oxidizing processes have already commenced. It is unfortunate that this body could not be inspected, for only a few specimens were available. The undetermined elements in this and the other samples usually consist of unimportant amounts of arsenic, magnesium, barium, aluminum, fluorine, chlorine, phosphorus, and antimony.

No. 2 represents an aliquot part of a number of shipments of sulphide from another orebody. It is very similar in mineralogical character and chemical composition to No. 1, except that it shows the results of the admixture of small amounts of wall rock, which lowers proportionately the metallic sulphide and raises the lime.

No. 3 is an aliquot part of several shipments of a sulphide ore of the low-lead class. This is the only known large occurrence of sulphides of this class in the district, though as an oxide it is the principal ore produced by several of the mines. The sulphur in the analysis more than satisfies the lead, zinc, and iron as galena, sphalerite, and pyrite, which is confirmed by the presence of these three minerals and the absence of pyrrhotite in all specimens of the ore examined. Arsenopyrite also occurs very sparingly, though not determined, and the gangue is composed principally of quartz and gypsum with minor amounts of mixed carbonates. The sample may also contain some oxides, as both sulphides and oxides are being mined, and in breaking the ore for shipment small mixtures are unavoidably made; but, except for the presence of the gypsum, this ore would fill most of the requirements of a sulphide of the silver-lead-zinc type, low-lead class. A large part of the deficiency in the total of the analyses is accounted for in the oxygen and water of the gypsum, which were not determined.

Analysis No. 4 is of a specimen of sulphide ore from one of the chimneys. Unfortunately, as is often the case with specimens, it runs unusually high in the most valuable metallic sulphide and low in gangue, but it is useful in conjunction with the others, as it so closely approximates a complete analysis. The mineralogic composition is pyrrhotite, small amounts of pyrite, a little oxide of iron due to exposure to the air, galena, sphalerite, and minor amounts of quartz and mixed carbonate.

No. 5 is an average of several samples of the deepest sulphide yet encountered, and is not far from being a normal average sulphide ore. Reconstructed, it is found to be composed of pyrrhotite, to the exclusion of pyrite, galena, and sphalerite, and a very little gangue. The sulphur in the average is a little low, but as this analysis was apparently made by rough commercial methods the discrepancy may be in the determination rather than in the ore.

These analyses are representative of the major part of the silver-lead-zinc sulphides now known in the district and considerable effort was expended to make them as reliable as possible; and while they leave much to

be desired, yet, used with care and discretion, they probably give as clear an idea of the primary ore of this type as can be obtained at the present time. It is to be hoped that in the future closer data may be obtained, especially with reference to the condition of the iron and the amount of sulphur in the unaltered ore. From these and similar analyses and a study of the records of several years' shipments of different mines, the following are offered as the probable analyses of the two classes of primary ores of the lead-silver-zinc type:

No.	Pb	Zn	Fe	SiO ₂	S	Mixed Carbon- ates	As	Total
1, Per cent.	13	11	32	4	33	4	1.5	98.5
2, Per cent.	3	6	20	20	30	15	1.	95.

The first analysis, or that of the high-lead type, is probably approximately correct. The ore is practically a pure mass of dark-colored, medium-grained, slightly friable sulphides, with very little gangue, and with sphalerite, owing to its relatively low specific gravity, and consequent relatively large volume, predominate to the eye. The three principal sulphides, galena, sphalerite, and pyrrhotite, in some cases are very intimately mixed; while in others the pyrrhotite occurs in pure masses an inch or more in diameter and with the lead and zinc also segregated, but to a less marked degree. As far as could be ascertained, the pyrrhotite appears to have been the earliest mineral deposited, followed by the sphalerite and then the galena, although apparently the period of the sphalerite deposition overlapped and continued throughout the period of the deposition of the lead sulphide.

A recalculation of this analysis gives: Galena, 15; sphalerite, 16.4; pyrite, 23.5; pyrrhotite, 34.1; quartz, 4; mixed carbonates, 4; and arsenic, 1.5 per cent. The latter occurs as arsenopyrite, but for the purposes of comparison with the resulting oxide has not been recalculated. The iron sulphide is arbitrarily divided in the above recalculation. Development appears to have proved that the occurrence of the iron sulphide as pyrrhotite or pyrite depends almost entirely on the depth at which the ore is found, and, as nearly all the high-lead sulphide known at the present time occurs in the deepest portions of the mine, it contains pyrrhotite to the practical exclusion of pyrite. Thus four of the five analyses given above are of the sulphide ores that come from the deeper horizons, and carry pyrrhotite rather than pyrite, and hence run lower in sulphur than the theoretical analysis which is supposed to be an average, not only of the known sulphides, but also of the ores now occurring as oxides. In this connection, it must be taken into account that unmistakable traces of primary pyrite occur in the oxidized orebodies in the higher zones, and it seems probable that while at a greater depth a slight

modification of the above analysis may be necessary, for the present developed ground it is correct.

The low-lead analysis is not worthy of much weight, as there are few opportunities for obtaining accurate data with regard to this class of ore, sulphides of the low-lead class being a rare occurrence. The body sampled consisted essentially of pyrite—to the exclusion of pyrrhotite—quartz, minor amounts of galena and sphalerite, and considerable mixed carbonates. As with the other class of primary sulphides, the principal economic value lies in the silver content, which varies too widely to give averages.

All the silver-lead-zinc orebodies now known have probably resulted from the oxidation of one or the other of these two classes of sulphide ores, and intermediate types have not been encountered, though, where leaching and enrichment have taken place, local occurrences of oxidized ores are found, which are only with great difficulty correlated with either of these two classes. It is believed that the two classes of ore were formed at about the same period, though perhaps not exactly at the same time, and, if so, the low-lead class preceded the formation of the high-lead ores.

The general source of both classes, as will be shown later, is probably the same, but the specific sources, or better, the channels along which the two classes traveled in reaching their present locations, are in the writer's opinion entirely distinct. It is not believed that the low-lead ore is a phase or stage in the mineralization of a certain area by the high-lead class, but that if the two types are ever found intermingled, yielding an ore having the characteristics of both classes, this will be due to the impregnation of an orebody of one type by solution carrying the other type, each coming through distinct and widely separated channels. In looking over the occurrences of these two classes of ore, it is found that to date there has been very little overlapping of the areas characteristically yielding either of these types. The low-lead class is found in the northern part of the main zone and to the east; the high-lead class covers the south and central portions of the area.

All primary ores are believed to have ascended from below along the North-South group of fissures, selecting a favorable channel and replacing the limestone of the walls until a body more or less elliptical in cross-section, called a chimney, was formed. It is readily shown in many ways that the orebodies are not the result of the filling of open spaces, but rather of replacement of the limestone, as, for instance, chert nodules are found in the primary ore unreplaced and in their original positions. The North-South group of fissures, as above pointed out, are parallel and were probably due to the folding of the strata, and the most productive areas occur near the axes of the anticlines or a short way down either limb. Usually the chimneys are intersected by strong cross fissures, but it cannot

be shown that these have determined the location of the chimneys, as they are usually later than the stage of ore deposition, and are so numerous that it would be impossible to locate a chimney in the developed area without its intersecting one or more of these strong cross fissures. The exact cause of the localization of these chimneys is not clear, though apparently a favorable point for such an occurrence is the intersection of two fissures of the North-South group of slightly different trends, as a N. 30° W. and a N. 10° W. or a North-South and a N. 10° W. It is probable, however, that the true cause of the placing of the chimneys at the particular points along the North-South fissures at which they are found will only be determined when these orebodies are worked downward to their ultimate source, as will be considered later.

The solutions made off from these chimneys at favorable horizons, usually along the North-South group of fissures, replacing the limestone and forming the horizontal manta bodies, already briefly described. In breaking off from the chimneys, the bodies show their maximum irregularities, for they make their way through the limestone with apparently little regard for fissures or bedding planes; usually are inclined upward away from the chimneys and trend toward the structural axes, but at a distance of from 100 to 200 m. they pick up a favorable horizon and some North-South fissure and follow these for great distances. Some irregularities, however, are always encountered, the bodies crossing from one North-South fissure to a neighboring one, varying in total width and height, and stepping up or dropping down a few beds, though much of this vertical movement may be due to undetectable faults. On the whole, however, one is struck more by the continuity and regularity of these deposits than by the minor variations.

Some of the important points in this simple deposition of the primary ores are believed to be: First, that the solutions came from below the greatest depth yet reached; second, that they rose along North-South fissures only at certain favored points, forming chimneys, not along the whole or any considerable length of the fissures as has been sometimes considered; third, that the solutions broke away from the chimney, usually though not invariably upward and outward; fourth, that, encountering a favorable horizon and a North-South fissure, these were followed more closely than would be expected, considering the uniform character of the limestone and the multiplicity of the fissures.

As the most important corollary to the above propositions, the following is offered: No primary ore or ore oxidized in place can occur without a definite connection, through an orebody of reasonable cross-section, to the ore-bearing channels that go to a greater depth than has yet been reached. This is not an entirely accepted principle, as it has been believed by many that the ore found in the mantas rose, when in solution, more or less along the whole length of the fissure which the manta follows;

that the solution deposited its burden when a favorable horizon was reached, thus forming the manta; and that the chimneys resulted from the intersection of the ore-bearing fissure with some cross fissure. To sustain this view, fissures of the North-South group are occasionally found 18 in. to 2 ft. in width for considerable distances and over some vertical range, filled usually with oxidized ores which, excepting for a few important differences, occur as would be the case if the whole length of the fissure were the channel through which the ore entered the mantas. The objections to this idea are that enlarged fissures of all groups are found below the oxidized orebodies; that, when followed downward, they eventually close up completely and die out without connecting with any other channel that may have been the source of the ore; that the filling always shows ore leached in as distinguished from ore oxidized in place; and that even these occurrences are the exception and not the rule, as in general there are few signs of open fissures of the pre-mineral type below the manta bodies, it often being impossible to distinguish any North-South crack whatever in the limestone 100 ft. below a manta, even after the minutest examination. This is the case with several of the very biggest and strongest mantas now known in the district.

Were it possible, as has been sometimes considered, to have an ore-body containing many thousand tons formed by deposition from a solution which traveled to this point along an almost microscopic crack, without leaving the slightest trace of mineralization to show that such ore-burdened solution had passed, then isolated bodies of ore could be found. It is the writer's contention, however, that it is not necessary to strain one's credulity and imagination to the acceptance of such a view, and that isolated bodies of primary ore or secondary ore oxidized in place do not occur in the Santa Eulalia district. In substantiation of this, nearly all known mantas can be traced decisively to the chimney which fed them, through continuous ore of normal cross-section, and where this has not been done, there is a field for development work.

The same principle is applicable to the chimneys to a less marked degree. While no one can say to what height above a certain point a chimney may extend, that is, how far upward it may make its way through the limestone, it is believed that true chimneys must extend downward beyond any depth yet reached toward the source of mineralization, perhaps eventually changing in composition and character, though little evidence of such a change has appeared.

The sulphide orebodies usually show a marked tendency toward horizontal bedding, owing to the replacement of the flat-lying limestone. This is particularly marked around the edges of the orebodies, where extensions, usually a few feet thick, are found making out into the walls on the more favorable beds, while in the chimney the unfavorable limestone beds sometimes lie nearly across the orebody, forming a false roof,

with a relatively restricted connection through them to the next favorable bed above. In an intermediate stage the slightly unfavorable beds are partly replaced by the ore, leaving large masses of pure limestone surrounded by the mineral matter. There is little gradation of ore into the limestone and the complete change from high-grade sulphide ore into almost pure limestone takes place within a few inches. Accompanying the sulphide ore in very rare cases are small open channels of circulation without mineralization, called dry caves, but these are clearly later than the ore and have not been found to date, except where the oxidation has commenced, and are apparently dependent upon the oxidizing solutions for their formation. Aside from these rare occurrences and a few vugs, measuring usually less than 6 in. across, open spaces visible to the eye are of rare occurrence in the sulphides of this type.

The Silver-Lead-Zinc Type Oxidized Ores

By far the greatest production has come from the oxidized ores of the silver-lead-zinc type; indeed, in comparison, all others are but interesting occurrences. To date, no unoxidized mantas of the silver-lead-zinc type have been found, so that the entire subject of these interesting manta bodies must be considered under the head of oxidized ores.

One of the things most desired from the practical standpoint of the development of new orebodies, and the following to ultimate ends of those bodies now known, is a clear, comprehensive knowledge of the process of oxidation in all its stages. Unfortunately, this is extremely difficult to obtain in the Santa Eulalia district. Our knowledge of the primary sulphides is limited; even the two theoretical analyses given above are open to criticism; and the changes which took place in the ore and the steps in the process of oxidation are poorly represented, the very slightly changed sulphide ores usually occurring in juxtaposition with completely oxidized carbonate ores, with the line of demarcation between the two sharply and distinctly drawn.

The only possible way in which this subject can be approached at present is to compare the normal sulphide with the normal oxide, leaving out the intermediate stages, and then consider the result of oxidation rather than the processes. The oxidized ores of the chimneys cannot be used in this comparison, for in the chimneys, as will be shown later, certain migrations of the metals have resulted in concentrations and impoverishments, so that, when the huge size and extent of these bodies are considered, the difficulty of arriving at the average character and content of the oxidized ore of the chimneys is at once apparent. In the mantas, however, other conditions obtain. The mantas lie practically horizontal, and, while the metallic content varies locally, the shipments of one month closely approximate the shipments of the year and these the shipments

of several years; and different mines on the same bodies, in spite of different methods of handling and mining ores, separating them into various products, etc., closely approximate each other. In fact, a compilation of yearly averages of several properties shows a variation of about 1 per cent. Pb, 1.5 per cent. SiO_2 , 1 per cent. Fe, 0.5 per cent. CaO, 0.5 per cent. Zn, 1 per cent. S, and 0.5 per cent. As, though the silver, as the principal value, varies more widely. These come from one chimney and its mantas, and show a sufficient uniformity for comparison with the primary sulphides of a similar type and class.

In the following analyses of oxidized ores No. 1 is of the high-lead class and No. 2 of the low-lead class:

No.	Pb	Zn	Fe	Mn	SiO_2	CaO	S	As	Total
1, Per cent.	13.9	1.8	28.5	1.5	19.6	6.1	1.5	1	73.9
2, Per cent.	4.4	1.5	10	1.5	19	23.8	1		61.2

Mineralogically recalculated No. 1 gives: Cerussite, 13.7; galena and anglesite (secondary), 3.8; smithsonite, 3.5; quartz, 19.6; sulphur, 0.5; gypsum, 2.7; calcite, 9.3; hematite, turgite, and limonite, 42.5; wad, 2.4; arsenic, 1; total, 99.0 per cent. The undetermined, besides the oxygen and carbon dioxide, are fluorine, chlorine, magnesium, barium, aluminum, phosphorus, antimony. Part of the lime should be figured as fluorite, though this occurs in almost negligible quantities; some of the iron should be with the calcite as an impurity; the form of the arsenic is indeterminate; but on the whole the recalculation is a very close approximation of the average composition.

The recalculation of the corresponding sulphides already given is: Galena, 15; sphalerite, 16.4; pyrite, 23.5; pyrrhotite, 34.1; quartz, 4; mixed carbonates, 4; arsenic, 1.5 per cent. (not recalculated). The theoretical specific gravity of the sulphide ore of this type and class is 4.8, while the true specific gravity of the ore masses more closely approximates 4.2, the difference being due to the porosity. The theoretical specific gravity of the oxidized ore of this type and class is 4.3, while the true specific gravity of the ore standing in the stopes is more nearly 2.4. The sulphide ore has then about 8 per cent. of void or open spaces, while the oxidized ore has about 44 per cent. of the same.

The number of tons of oxidized ore resulting from a given number of tons of sulphide ore is unknown, as are the relative volumes occupied by these ores, as will be shown later, so that there is but one basis on which a comparison of the two can be made, and that is on some element which remains constant in total in a given section. The lead apparently is the only metal which answers the purpose. It is not readily soluble, is relatively immobile, and if found outside of the walls of the orebody usually contains a gangue, which, averaged over a number of cases, closely approximates the general gangue of the ore; if it is partly removed by the

oxidizing solution from the original site of deposition, it is removed proportionately to the other elements, thus not interfering with the relative values obtained by the comparison of the two types.

Comparing the two analyses, we find that there is slightly more lead per ton in the oxidized ore than in the sulphide, hence the number of tons has decreased a very little during oxidation, and the proportion between the tonnage of the sulphide and the tonnage of the oxide is as 13.9 to 13. Applying this corrective factor to the comparison, we get the actual loss and gain per metric ton of primary sulphides during the oxidation process as follows: Loss, 93 kg. of zinc, 316 kg. of sulphur, 6 kg. of arsenic and 53 kg. of iron, making a total loss of 468 kg.; gain, 144 kg. of silica, 71 kg. of impure calcite and gypsum, 45 kg. of carbon dioxide, aside from that contained in the calcite, and 142 kg. of oxygen, making a total gain of 402 kg. The resulting orebody contains almost the same number of tons of ore as did the primary body, but the secondary ore, owing to its lower specific gravity, occupies a much larger space. While the above is open to criticism in the analyses themselves, still it is believed, not only that it is a close approximation of the truth, but also that in no other way can a conception be gained of the actual results of the oxidation process.

If it were possible to obtain an analysis covering an equally large amount of the low-lead class of ore, this could be treated in the same way, but, unfortunately, both the sulphide and oxide analyses given for this class of ore have not sufficient weight to warrant mathematical deductions.

The oxidized ores, particularly the high-lead class, are predominately soft, friable masses of siliceous iron oxides and lead carbonates, loosely massed together and extremely porous. Though all possible variations and combinations occur, a typical cross-section of an oxidized manta orebody is about as follows: The roof is relatively flat at the center, being nearly always a bedding plane of the limestone; 40 or 50 ft. below, the floor is rounded, with perhaps a rapidly diminishing extension downward along some fissure; the walls are 100 to 150 ft. apart, and one or both are often irregular, with mantas 5 or 6 ft. thick extending out into the limestone a few meters. On the roof is a light yellowish brown, extremely finely divided earthy deposit several inches thick, composed principally of silica and alumina with a little magnesium carbonate, manganese, and iron oxide. Below this is an open space varying from a few inches to 10 or 15 ft.—in an orebody with the dimensions given, being often 4 or 5 ft. deep—and extending clear across the orebody. This open space may often contain beautiful growths of calcite, stalactites and stalagmites, and fine branching intergrowths of calcite tubes, varying from 1 to $\frac{1}{64}$ in. in diameter, with those under $\frac{1}{4}$ in. in diameter hollow and built up by lime-bearing water traveling along the interior. In many cases, however, these calcite crystals and growths are lacking.

Below is the surface of the ore, sometimes with a few boulders of limestone fallen from the roof, making an irregular surface, on which is almost invariably a coating several inches thick of gypsum which can best be described as closely resembling snow. It is as soft as snow, has a slight crust easily broken, can be pressed into balls in the hands, and lies on flat or gently sloping surfaces, and not on steep ones. Sometimes a little wad or pyrolusite is found in places lying above the gypsum, while below is often a layer of from 2 to 6 ft. of nearly pure cerussite or lead carbonate, bluish in color and so soft as to be readily shoveled without picking. From this point downward, the ore is a mixture of siliceous iron oxide and lead carbonate, lying loosely on the limestone floor. Along the walls are found the little galena and anglesite that occur with the ore, and where they are present in considerable amounts these secondary sulphides extend out into the walls as solid mantas several feet thick.

In the vertical chimneys of the high-lead class portions containing little lead are common, while just above the sulphides at one point a mass of pure cerussite 50 ft. thick and the full size of the chimney was found; a result of concentration of values in these vertical bodies. Large open spaces are common in these oxidized chimneys, and where an unreplaced bed of limestone extends nearly across the body an open space occurs just below it, similar to that found below the limestone roof of the manta bodies.

These open spaces have been attributed to "shrinkage," on the supposition that, due to the loss of sulphur and zinc, the ores must have lost in bulk during the oxidation, and this term certainly describes the appearance of the open spaces admirably. However, in the consideration of the results of oxidation it was shown that the decrease in a total number of tons is so slight as to be almost negligible, and that, owing to the lowering of the specific gravity of the oxides and the increase in porosity, the oxide ore occupies a greater space than the sulphides, the ratio being inversely as their specific gravity or as 42 to 24. Locally, portions of the entire ore-body have been removed into cross fissures. These figures are based on the ore alone and include porosity, but do not include the open spaces (or "shrinkage") which occur in the oxides but not in the sulphides. This factor, then, further increases the size of the carbonate orebody, so that its cross-section from limestone to limestone must be increased during the oxidation process at least one-third in all cases and probably often one-half. This accounts for the discrepancy in size between the oxides and sulphides noted in some of the chimneys, for the cross-section of the sulphide or deeper portion is very much smaller than the cross-section of the oxidized chimney above, which, under the theory of "shrinkage" of carbonate ores, caused some apprehension as to the prospects of the district with deeper mining. Such apprehension, however, is falsely founded, for, when carefully considered, it is found that the tonnage remains the

same, if anything slightly increasing with depth, it being only the distance between the limestone walls that has decreased with the change in character from oxides to sulphides.

It is apparent that such increase of size of the ore-bearing cavity must be due to the solution of the limestone wall rock of the orebody during oxidation; and a consideration of the probable process of oxidation shows that undoubtedly such action has taken place. The sulphide with its four principal metallic minerals, pyrrhotite, pyrite, galena, and sphalerite, is acted upon by oxygen-bearing waters which also at some stage carry considerable silica. The iron sulphides were first attacked, probably yielding ferrous sulphate and sulphuric acid; the former then passed to the hydrous oxide with the generation of more sulphuric acid, and very probably with the intermediate step of the formation of the ferric sulphate, itself known to be a strong oxidizing agent. The sphalerite is readily attacked by these iron salts and the free sulphuric acid, but, although its great mobility and ease of solution are well shown at Santa Eulalia, it apparently does not commence to oxidize until the oxidation of the iron sulphides is well under way. It certainly formed zinc sulphate as a stage of the oxidation, but goslarite is found only sparingly as an efflorescence on the limestone, while the usual deposit along channels of circulating waters is the carbonate, and it seems probable that much of the zinc was carried away from the orebody in solution as zinc carbonate, after the reaction between the zinc sulphate and the calcium carbonate had resulted in gypsum and smithsonite. Hydrozincite is not rare at Santa Eulalia, but its relation to the process of oxidation could not be determined. The oxidation of the lead follows slowly and there is abundant evidence that the galena is attacked strongly only when intimately mixed with the other more readily oxidized sulphides, due probably to the generation of sulphuric acid and sulphates of iron in the mass of mixed ore, and to the removal of the more soluble sphalerite, with the consequent development of a cellular porous structure in the galena. When recrystallized as secondary galena, in which form a small amount of lead occurs in all orebodies, it is free from iron and zinc sulphides, and in this pure state is almost stable. It then apparently undergoes oxidation with extreme slowness; much of it remains unchanged, but a fair proportion goes over into the sulphate, anglesite, and this also, when pure, appears to be practically stable, yielding only a slight coating of the carbonate, cerussite. This marked difference in solubility between the primary and secondary galena appears to be due, first, to the differences in physical character, the primary being cellular, while the secondary, being massive, is only attacked on its exterior surfaces; and second, to the fact that the main processes of oxidation having been completed, the generation of sulphuric acid and the formation of active ferric sulphates practically ceased before the reoxidation of the galena commenced. The sulphate, anglesite, may

be deposited in considerable quantities for the first time during the oxidation of the secondary galena, for, although the formation of the lead sulphate is believed to be a step in the alteration of the galena to cerussite, it is not commonly found in Santa Eulalia as an alteration product of the primary ore. Apparently the reaction between the lead sulphate and the calcium carbonate took place immediately upon the formation of the sulphate, and the process was recorded only by the deposition of the cerussite. This same stability is noticed in the third or last stage of primary mineralization, the narrow galena seams extending up through the porphyry to the surface. Although these seams outcrop, they are unaltered.

In many of these processes free sulphuric acid is liberated, at least a portion of which probably attacks the limestone walls of the orebody, forming gypsum and enlarging the sides of the chamber, while the sulphate radical acts similarly in the double decomposition of the lead and zinc sulphates with the calcium carbonate. A comparatively insignificant part of the gypsum thus formed remains as a coating on the orebody, as above noted in the description of the typical cross-section of the average manta, but by far the greater part of it is carried off with the uncombined sulphuric acid in the circulating waters. The insoluble portions of the limestone, usually a very small percentage of silica and a little clay, remain as a finely divided deposit in part on walls and roof, but principally in the deepest parts of the floor of the orebody along the channels of circulation.

Thus the orebody, which as a sulphide was strictly a replacement of the limestone, as an oxide takes on the characteristics of a deposit formed in a solution chamber.

During the oxidation, a certain amount of readjustment of the metals remaining in the orebody took place. Apparently the principal reaction between the solutions and the limestone occurred along the roof of the orebody, as shown by the bed of pure cerussite, just under the gypsum, and by the residual clay deposited against the roof; considerable action took place along the walls also, where the secondary galena and accompanying anglesite are found almost exclusively.

In general there is abundant evidence that the action of the oxidizing solutions following the open channels has been downward, and indications are found in the water channels that the more migratory minerals, smithsonite, gypsum, etc., passed along these channels in that direction. Mineralization occurs in cross fissures underlying the orebodies from which it was derived, and with which it is often connected, and while iron and manganese stains are the rule below the oxidized orebodies, fresh unstained limestone is the rule above. These observations apply to the mantas of both classes rather than to the chimneys, for the latter offer open channels for circulating waters.

There is in the manta bodies, moreover, no secondary enrichment in the sense ordinarily used in deposits in veins where minerals are leached out of upper portions and reprecipitated in lower portions of the same vein, because first, in the main zone the average oxidized manta occurs at a depth of about 800 ft., very few mantas are found even locally at less than 200 ft., and the capping itself is often 400 or 500 ft. thick; second, the solutions that pass through one manta body do not necessarily encounter any other orebody in their downward course; third, silver, the principal economic mineral, seems to remain with the lead and hence usually stays in the orebody; fourth, no definite water level has yet been encountered, although several shafts have found a very small flow at about 1,500 ft.

A typical example of an orebody in a cross fissure or an orebody extending downward along the main fissure, due to the action of oxidizing solution, shows: First, the normal orebody or source; then, usually narrowing rapidly, practically normal ore carrying lead carbonate; below this, iron oxide carrying no lead; zinc carbonate grading into a mixture of fine siliceous clay and hydrozincite; and then a tight fissure. One such occurrence measures an average of 40 ft. across, and each of the stages shows nearly 100 ft. of vertical range. It will be noted that these minerals occur in the normal order of their solubility, the most soluble traveling farthest from their source and the finely divided clay, probably carried practically in suspension, collecting at the bottom. Such an ore occurrence is nearly complete in itself; that is, every constituent but a portion of the calcium carbonate and the gypsum being accounted for. The fissure, therefore, offers no inducement to prospecting downward except the possibility of discovering another independent body. At least one such orebody is known in each of the principal mines of the camp; nevertheless this is not the usual occurrence, for normally there is no obstruction in the channels to oxidizing solutions and the more soluble products are carried beyond the range of observation and cannot be completely accounted for.

As already shown, there was a loss, during oxidation, of 93 kg. of zinc per ton of primary ore, which is approximately a ton of carbonate ore; and when the total production of the carbonate ores to date is considered it is at once apparent that the circulating oxidizing solutions have handled an enormous tonnage of oxidized zinc minerals. To date, with the exception of such minor occurrences in cross fissures and the extension of orebodies downward as mentioned above, very few zinc bodies have been found, and the tonnage produced from these amounts is but a fraction of that yet to be found. As compared with other districts, the oxidation in Santa Eulalia was characterized by plentiful amounts of water, usually, though not always, circulating with comparative freedom, and by a deep water level not yet reached. In deposits in limestone, where

the water is largely confined to the orebodies, the smithsonite and calamine resulting from the oxidation are often found in the orebodies themselves, deposited along the walls and especially along the floors of the mantas, runs, blankets, flats, or whatever they may be called. Where the oxidation has been effected by barely perceptible trickling waters in a dry mine, zinc is often found in the fissures and caverns in the neighboring limestone, in crusts and veinlets sometimes but a few inches thick deposited directly from the supersaturated solution, as smithsonite and calamine; but these minor occurrences in the aggregate account for the total zinc of the unoxidized ore bodies. Where the amount of water was great and the circulation was free, as was evidently the case in Santa Eulalia, the zinc minerals were carried far from their source, and the channel through which the unsaturated solution passed may not contain even a trace of the oxidized minerals, and may simply show that a large amount of solution of some sort has passed through it.

The problem is not a simple one, owing to the multiplicity of channels in the limestone; but in some cases it has not received the attention which the value of possible positive results should warrant. It is not easy to recognize the zinc minerals resulting from oxidation, smithsonite alone having an almost endless variety of forms, many of which are extremely inconspicuous, and it is probable that only by continued systematic sampling for zinc can the problem be attacked with any degree of certainty.

The plotting of such zinc assays gives some interesting results. On one of the large mantas of the district this was attempted and it was found that a curve could be drawn which showed by its variation the location of the channels of circulation, for where the manta passed over a downward extension of the ore the zinc content was low, rising with increasing distance from this point and falling again where another downward extension or an open fissure was passed. If these downward extensions are true chimneys, the zinc content does not rise in them until the level of the sulphides is approached, at which point a body of high-grade zinc ore sometimes occurs in the limestone near the chimneys. On the other hand, if the extension downward is merely a local enlargement, the zinc rapidly increases with depth, and the clay content increases, under which conditions further exploration downward is inadvisable at that point. Thus the recording of zinc assays not only assists in the locating of bodies of zinc ore, but also shows clearly the channels of circulation of the oxidizing waters and those points at which prospecting should be done to determine the downward extension of the orebodies.

As will be readily realized, these conditions are capable of endless variation and the data must be carefully taken, thoroughly studied, and correctly interpreted, in order that a sufficient benefit may be derived to repay the labor involved in its compilation.

One of the best indications that a water course, channel, or fissure has been the locus of oxidizing waters that have acted upon and assisted in oxidizing an orebody, is the presence of a deposit of gypsum along it. This mineral also travels great distances from its source and is deposited in the limestone in caves or caverns, lined with beautiful white crystalline deposits of the mineral, and is sometimes spread out as an impregnation of the limestone itself. Such fissures or channels are favorable sites for the deposition of zinc carbonate or silicate.

"Painted limestone," or limestone stained with dark manganese oxide and pink carbonate and silicate, has long been recognized as a favorable indication of the vicinity of oxidized ore. Indeed, in a few instances it carries sufficient silver to be an exceedingly valuable ore in itself, and one highly prized by the smelter. Such ores occur, first, where the orebodies are relatively high in silver; second, where there is a special inducement to impregnation of the limestone, as, for example, where a chimney has jumped from one fissure to another, leaving a vertical oxidized porous chimney of ore practically resting on limestone; third, where the limestone for physical reasons is favorable to impregnation by descending solutions. In general, manganese-stained limestone is not necessarily silver bearing, and occurs to some extent along all fissures which have carried products of oxidation, except where these are encountered at great distances from the source of the oxides and consequently the solutions have been divested of all but the most soluble minerals. Gypsum and zinc compounds are thus found along a fissure or channel long after the manganese has ceased to be present. Normally the manganese-stained limestone occurs beneath the entire orebodies, never above, excepting along some fissure that carries oxidation products of some other orebody still above, or along the channel of the deposition of the third or pure galena type of mineralization.

The amount and intensity of this "painting" is an indication of the distance from the ore above. Were it not for a few drawbacks, this "painting" would be an invaluable aid to prospecting for oxidized manta bodies. As it is, it is merely a helpful indication to be carefully applied. The "painting" is often noticeably lacking, particularly in the vicinity of strongly marked open cross fissures, where the solutions have followed these open channels instead of impregnating the limestone mass. Ore may thus be found overlying unstained limestone. Certain horizons in the district, unusually favorable for deposition and in which several orebodies have been found, are "painted" by manganese carbonates and oxides over great areas, and it is then a question whether impregnation took place during the primary deposition, during the oxidation, or whether minute traces of manganese were deposited coincidentally with the limestone itself. Very often this type of "painting" occurs immediately below beds of fossils, and, while not continuous over the whole district, it

certainly is much more widely distributed than the orebodies. It has never been found associated with the sulphide ores and, if present below the level of oxidation, it is either very inconspicuous or the proper horizon has not yet been encountered. The great extent of this class of "painting" tends to mislead rather than to assist in the development and location of orebodies.

The origin and source of the large amounts of silica added during oxidation to all known occurrences of the high-lead class of ores, and to some occurrences of the low-lead class, are not clear, nor do they appear readily determinable. The channels in the limestone along which the oxidizing waters have passed are not silicified, nor are any large occurrences of silicified limestone known in the district. The capping, however, has been silicified over extensive areas and it is but natural to conclude that the same solutions effected the silicification of the orebodies. The most illuminating occurrence is that found in one of the deposits of the low-lead class, where the chimney stands for nearly 800 ft., running 20 to 30 per cent. higher in silica than the normal primary ore of this class and than some of the mantas. Where the chimney joins the enormous upper mantas the highly siliceous ore appears to be present in chutes, conveying strongly the impression that it was introduced after the period of oxidation, and that it followed channels through the ore rather than penetrating the whole mass. It seems probable, from our present limited knowledge, that the silica was introduced at about the same period as the third or pure galena type of primary ore deposition, but whether simultaneously, just before, or just after, is not certain. It is probable that it came from below; that it followed the oxidized chimney as a channel; that it continued upward into the overlying capping; and that it represented a distinct stage which has now passed.

The oxidized ores present an unlimited number of interesting occurrences, each illustrating some point or factor in the study of these deposits and each requiring separate investigation and a separate description, but the general principles governing the deposition are believed to be essentially those given.

"Contact" Bodies.—One of the points on which there is much difference of opinion among the operators of the Santa Eulalia district is the relative ages of the mineralization and the tuff, and the value of the contact as a locus of orebodies. The latest phase of primary mineralization, the third or pure galena type, resulted, as already described, in narrow vein-like streaks of galena, free from iron sulphide or sphalerite, usually about an inch in width. These narrow galena seams are found throughout the limestone mass, but only in the upper horizons thus far. They extend up into the overlying beds of tuff, usually on N. 30° E. fissures, thus showing that this tuff, where fissured, is capable of mineralization and that the capping was present when this last primary action was taking

place. Had the capping been present when the main silver-lead-zinc ores and silver-bearing pyrite ores were being deposited, traces of these mineralizations would probably be found in the capping.

The dikes cut both the orebodies and the capping and, as has been pointed out, were later than the primary ore and earlier than the oxidation. Hence the capping was present during practically all of the great period of oxidation and its accompanying ore movements and was a considerable factor in controlling the circulation of the oxidizing waters.

The relative ages of the tuff and the mineralization cannot be absolutely determined by the study of the various fissures and their ages, but the results are not without interest in this connection. No definitely certain pre-mineral fissures continue upward into the overlying tuff and, of the earlier post-mineral fissures, only those that have strongly marked later fault movements do so, while the latest fissures outcrop characteristically on the surface of the tuff, often having marked effect on the topography and sometimes, as noted, carrying mineralization of the third or pure galena type into it. It is conceivable that fractures may have occurred in the limestone, continued up to the tuff, and died out without entering even a few inches into the overlying rock, but it does not appear probable; and it seems rather that the study of the fissures in this connection places the age of the tuff as later than the pre-mineral fissures, later than the first post-mineral cross fissures, and earlier than the last post-mineral cross fissures. Hence both of the economically important types of ore deposition appear to antedate the capping.

The oldest type of silver-bearing sulphide ore occurs directly under the tuff at one point where the ancient surface was low, and it is scarcely conceivable that this type of mineralization could take place at the shallow depths (probably at no time exceeding 1,500 ft.) represented by the overlying capping. Certainly, if this were true, a reconstruction of one of the basic ideas of economic geology—that the minerals are a criterion of the depth at which deposition took place—would be necessary. For the same reason it does not seem possible that the lead-silver-zinc bodies of such great vertical range could have formed at such a relatively shallow depth. (See p. 195.)

If the tuff had been present when the ore was rising along the chimney, action resembling that found in similar deposits in a region where shale is interbedded with the limestone or overlies it would be expected; that is, there would be a widening out or spreading of the deposit just below the impervious beds, particularly developed where the shale or tuff was practically horizontal, or better, where it formed a large dome. Such occurrences are common in the lead-silver-zinc deposits in many districts, but nothing of a similar nature occurs at the contact between the tuff and the limestone in Santa Eulalia.

The known occurrences in the main zone at Santa Eulalia are illus-

trated by three orebodies partly or completely worked. These bodies are all similar in that they are rather small and erratic, yielding but a small fraction of the ore produced from the average manta, and this is often concentrated in caves or bodies with relatively little ore between. In no case do they follow a North-South fissure or any other system of fissures for any great distance, but break from one set to another, apparently following the most promising channel, regardless of direction. The contact is in each case steeply inclined, and the action, where it could be determined, appears to have been downward along the course of the fissures rather than upward. There is no tendency to spread out along the contact; in fact, the ore very often is not in contact with the tuff, but from 5 to 30 ft. below, in the limestone, though always touching the sloping contact at intervals. Occasionally some impregnation occurs in the tuff, but this, as nearly as can be ascertained, took place as an oxide, not as a sulphide. The ore itself varies considerably; one of these bodies belongs to the low-lead type, the other three to the high-lead class, but only in one of them was the lead present in appreciable quantities.

It seems probable that these bodies were deposited in their present location as oxides, the ore-bearing solutions working downward along the most favorable channel, in this case that formed by the intersection of fissures of all ages with the contact itself; and that the source of the ore was in all cases manta orebodies which lay normally in the limestone, just as the known orebodies lie to-day, but which outcropped on the old surface before the tuff was laid down, just as several mantas outcrop in the cañons of the present topography.

The mantas lay practically horizontal, as sulphides, while the erosion was going on, the climate of that time in Mexico undoubtedly being such that erosion took place as rapidly as oxidation. Certain of the orebodies were exposed on the surface of the steep hillsides, particularly on those slopes running approximately east and west, or at right angles with the general north-south trend of the orebodies, while in other cases they were completely removed, as evidenced by the indications on the surface at certain points. Those bodies exposed on the hillside and running back into the limestone were buried again under the showers of tuff or volcanic ash and were later oxidized. As the whole cross-section of the orebodies was exposed at the sloping contact or old surface, a natural channel for circulating waters, it is not remarkable that great quantities of oxidized ore were carried down along this channel and deposited in it. Two of the known contact bodies are believed to have resulted from one of these mantas since completely removed by erosion, and in consequence the contact bodies themselves now outcrop on the surface. In other cases the contact bodies, if followed upward, would undoubtedly lead to a manta in the limestone at a higher horizon, and a case is conceivable

where, after following such a manta for some distance, the contact would be encountered again and a second contact body similar to the first be found pitching down the other slope.

There is one case in the district where a strong manta following definitely a North-South fissure, lying at a short distance below the contact and in places touching it, has all the characteristics of a typical manta orebody, in size, in cross-section, oxidation products, etc.; it lies practically horizontal and follows a certain bed of limestone that is apparently favorable for deposition of ore. This is not a contact body and the fact that the contact is sometimes just above and always within 100 ft. has nothing to do with its occurrence; it seems more probable that the favorable limestone lying just below the tuff, which is associated with it, is the determining factor. In substantiation of this view, other bodies are known in Santa Eulalia which lie for some distance at a scarcely greater depth below the present surface; given a slightly longer period of erosion and another deposition of tuff, the same sort of occurrence as that just described would result.

It is believed, then, that the tuff was laid down after the principal period of primary mineralization, and certainly after all primary mineralization of economic importance; that it antedates the great period of oxidation and accompanying readjustment; and that the limestone just below the tuff is as favorable for oxidized manta orebodies as other limestone horizons, but not more so, unless for some reason independent of any consideration of the overlying tuff. Also, that the contact, where steeply pitching to the north or south, is a favorable location for minor occurrences of secondary ores, which are economically important principally as they may lead upward into the oxidized manta at a higher horizon of the limestone.

Probable Source of the Ores

We have, thus far, in a brief and incomplete manner considered the primary deposition of the three types of ores, their occurrence and their process of oxidation. There remains only the consideration of their ultimate source, a subject dealing purely with horizons not yet reached in Santa Eulalia, and conditions that can be only conjectured.

It is evident that at Santa Eulalia the two economically important types are far more distinct than is usually the case with the variations found in a single restricted area and a single formation. The silver-bearing iron sulphide type is distinct from any other known deposits in its mineralogical association of ilvaite, fayalite, knebelite and magnetite as gangue minerals, and silver-bearing pyrrhotite and pyrite and arsenopyrite as metallic sulphides. The development of the type is not complete and it is very possible that with the present 1,000 ft. of vertical range more fully exposed, conclusive evidence may be obtained as to

the manner of its formation. It undoubtedly belongs to the class of deposits illustrated by the Ducktown, Tenn., and Granby, B. C., copper mines; that is, deposits resulting from igneous metamorphism from a source not yet exposed. The relation of the feeder fissures and the shape and occurrence of the orebodies seem to point toward a source lying vertically below.

The later type of silver-lead-zinc ore at Santa Eulalia is remarkable for its great vertical range, having been traced through 2,000 ft. of limestone without appreciable change in metallic content. It is usual, in deposits of this type, to find the zinc and particularly the iron sulphide increasing with depth, resulting in an ore that resembles the metallic sulphides of the silver-bearing iron sulphide ore of Santa Eulalia, but to date such a change is not apparent.

These characteristics would seem to point to the formation of both types of deposits at unusually great depth and to an exceedingly protracted period of mineralization during a very slowly falling temperature. Lindgren points out that the silver-lead-zinc type may be deposited at depths exceeding 12,000 ft. (*Mineral Deposits*, p. 513 [1913]), and the gangue minerals of the silver-bearing iron sulphide type of Santa Eulalia are normally higher-temperature minerals than the silver-lead-zinc type. It seems probable, then, that the deposits at Santa Eulalia were formed at least 10,000 ft. below the surface and very possibly at much greater depth.

Approaching the subject from a consideration of other mineral occurrences lying in the same Cretaceous formation and in the same metallographic province, some additional ideas are derived. J. E. Spurr and G. H. Garrey have studied two of these deposits in great detail,⁵ and from these and similar occurrences Spurr was able to arrange the widely varying types of mineralization in a column showing that all these represent progressive changes in the mineralizer which has its source well within the igneous rock, and that this mineralizer is essentially but a magmatic phase. He has given⁶ seven zones, or stages, arranged progressively, of which stage C—cupriferous pyrite, stage D—argentiferous pyrite and auriferous arsenopyrite, stage E—zinc blende, and stage F—argentiferous galena, are extremely common in northern and north central Mexico, throughout the Cretaceous limestone. The writer has had on opportunity to study in some detail both occurrences described by Spurr and Garrey, since the appearance of these articles, as well as numerous other

⁵ J. E. Spurr, G. H. Garrey and Clarence N. Fenner: *Economic Geology*, vol. vii, No. 5, pp. 444 to 484 (Aug., 1912).

J. E. Spurr and G. H. Garrey: *Economic Geology*, vol. iii, No. 8, pp. 688 to 725 (Dec., 1908).

⁶ *Economic Geology*, vol. vii, No. 5, p. 489 (Aug., 1912).

deposits in the same metallographic province, and, while probably some slight changes in the grouping or subdivision may be necessary in the future as they become better exposed by further development, the general idea admirably fits the ore occurrences of this section. One of the difficulties that stand in the way of its ready application to practical problems is that the arrangement was necessarily made according to two factors rather than a single one: first, according to the physical conditions, or the decreasing temperature and pressure as the distance from the seat of igneous activity increased and the distance from the surface decreased; and second, according to the time of deposition or the gradual changing of the mineralizer and its metallic contents as time progressed, independently of the temperature. Either of these factors may cease in its normal progression; for instance, the recurrence of igneous activity after any number of stages have been passed may start afresh the entire series, or intense faulting, or the extrusion, as dikes or flows, of large masses of igneous rock may at any stage in the process so reduce the physical conditions of temperature and pressure over local or extensive areas as to allow the deposition of the remaining stages to be superimposed over the earlier. Thus one case has been noted where a fault of great magnitude occurred after the conclusion of the "C" or cupriferous pyrite stage, and bodies of the "E" and "F" or sphalerite and galena stages were found largely along the fault zone and slightly impregnating the orebody, it being as yet unknown whether the deposition of the "D" or argentiferous pyrite and auriferous arsenopyrite stage took place before or after the faulting, though this knowledge is of primary practical importance, as in the first case stage "D" would lie above the copper stage and has then been removed by erosion, and in the second case it would lie below the sphalerite and galena stages and hence must be looked for with depth.

At Santa Eulalia we have exposed the "F" or argentiferous galena zone in the third or latest type of mineralization, though only as an unimportant mineral occurrence; "F" and "E," or lead and zinc stages, strongly overlapping, as these stages often occur in the silver-lead-zinc type or main ore occurrence; and "D," the new, more recently discovered silver-bearing iron sulphide type, containing argentiferous pyrrhotite and pyrite and slightly auriferous arsenopyrite, and carrying as a gangue iron-bearing contact-metamorphic minerals belonging to a still deeper stage. For some reason, similar to those just outlined, the stages overlap. It is probable that as depth increases we will find increasing amounts of metamorphic minerals, these changing gradually to the deeper-seated types; and with this change will come, possibly, increasing amounts of copper-bearing sulphides, and finally the igneous rock, probably a monzonite or one of similar composition. When this rock is encountered and explored, we will probably find that the cause of the location of the silver-

lead-zinc chimneys is due to irregularities in the shape and occurrence of the igneous rock rather than to intersecting fissures, etc. The depth at which this igneous rock may be encountered cannot even be estimated. Spurr found at Dolores nearly a mile of vertical range covering only portions of the argentiferous pyrite-auriferous arsenopyrite zone and the cupriferous pyrite zone, showing that under favorable conditions one of these stages may have a great vertical range. It seems probable that at Santa Eulalia, as the lime-iron silicates have already appeared, the distance to the igneous rock cannot be great; still the thickness of the limestone is unknown and the igneous rock will probably not be a factor in the development of the district for many years.

Summary

In the following chronological table, the principal events in the formation of the Santa Eulalia ore deposits are arranged as they appear to have taken place.

1. Limestone laid down in Cretaceous sea.
2. Land surface uplifted at close of Cretaceous, with little development of structure.
3. Hypothetical igneous mass intruded; slight structure in limestone developed.
4. North-South class of North-South group of fissures formed.
5. Magmatic processes set up at some point in the mass of igneous rock, accompanied by the giving off of mineralizers which formed hypothetical contact-metamorphic deposits at some depth below that yet reached.
6. Silver-bearing iron sulphide type of ore deposited along North-South fissures.
7. Other classes of the North-South group of fissures formed.
8. Silver-lead-zinc type deposited.
9. N. 55° W. fissures formed.
10. N. 70° W. fissures formed, followed by a long period of active erosion of the limestone surface perhaps amounting to 10,000 ft. or more.
11. Capping of tuffs and volcanics laid down.
12. Andesite and rhyolite dikes intruded.
13. N. 30° E. fissures formed.
14. Oxidation on a large scale commenced.
15. Silicification of much of the oxidized ore and overlying capping took place, with mineralization of the third, or pure galena, type just preceding or following it.

Acknowledgments

The writer is indebted to the Managers and Superintendents and staffs of the mines of the district for many kindnesses, every opportunity having been afforded him for the gathering of the data upon which the paper is based, and wishes particularly to acknowledge his indebtedness to A. L. Eaton, Superintendent of the Chihuahua and Potosi mining companies, for much helpful information.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Petroleum Fields of Alaska*

BY ALFRED H. BROOKS, WASHINGTON, D. C.

(New York Meeting, February, 1915)

Introduction

PETROLEUM seepages are known in Alaska at four localities, all on Pacific seaboard. These, named from east to west, are Yakataga, Katalla on Controller Bay, Iniskin Bay on Cook Inlet, and Cold Bay on the Alaska Peninsula. Besides these, a petroleum residue has been found near Smith Bay on the north Arctic coast of the Territory. At Katalla, Cold Bay, and Iniskin Bay there has been some drilling for oil, and in the first field several productive wells have been opened up. Alaskan petroleum, so far as its composition is known, is a refining oil, with paraffin base and low sulphur content.

Most of what is known of the geology of Alaska oil fields is based on the investigations of Dr. George C. Martin, of the United States Geological Survey. A. G. Maddren, of the same service, has recently made a reconnaissance of the Yakataga oil field. The data to be here presented are chiefly taken from the following publications:

G. C. Martin: The Petroleum Fields of the Pacific Coast of Alaska, with an account of the Bering River coal deposits, *Bulletin No. 250, U. S. Geological Survey*, pp. 9 to 27 (1905).

G. C. Martin: Geology and Mineral Resources of the Controller Bay Region, Alaska, *Bulletin No. 335, U. S. Geological Survey*, pp. 112 to 130 (1908).

G. C. Martin and F. J. Katz: A Geologic Reconnaissance of the Iliamna Region, Alaska, *Bulletin No. 485, U. S. Geological Survey*, pp. 126 to 130 (1912).

A. G. Maddren: Mineral Deposits of the Yakataga District, *Bulletin No. 592, U. S. Geological Survey*, pp. 143 to 147 (1914).

Katalla Field

The Katalla field is marked by a series of seepages and gas springs occupying an east and west belt, about 25 miles in length and from 4 to 8 miles wide. (See Fig. 1.) This zone skirts the north shore of Con-

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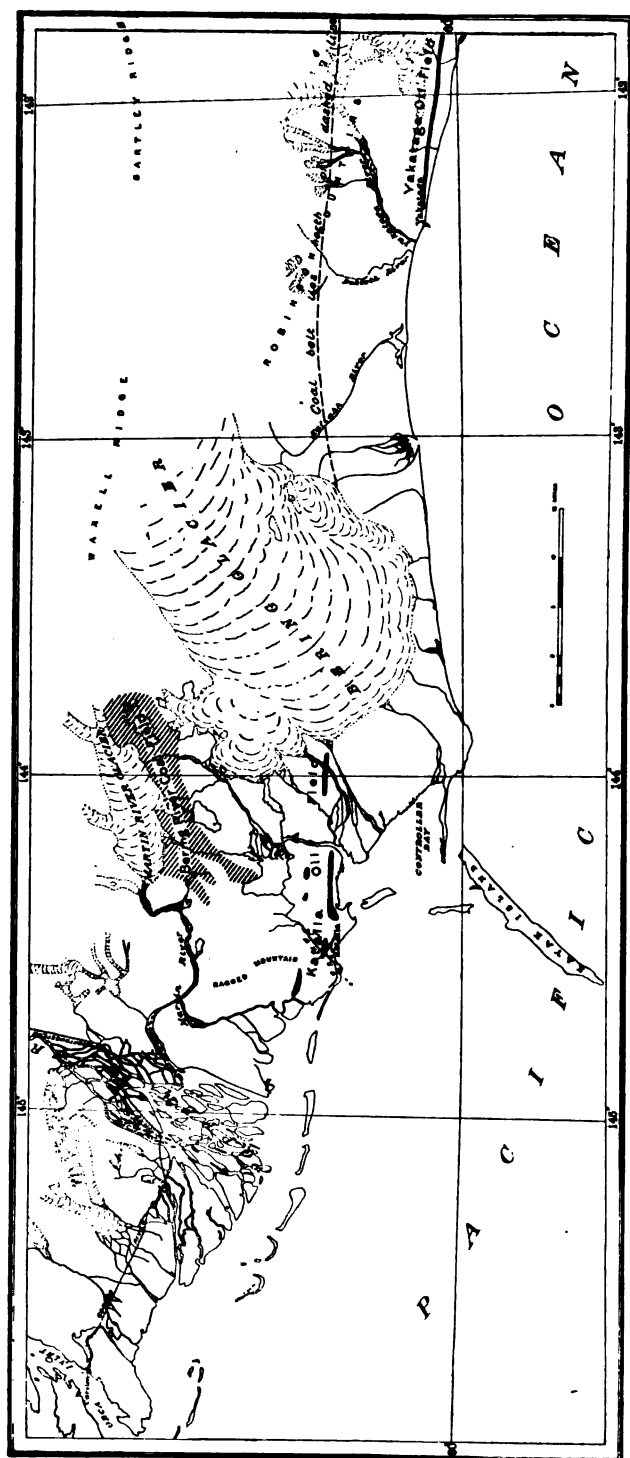


Fig. 1.—Map showing location of Katala and Yakataga oil fields, Alaska

troller Bay. To the east it extends into the alluvial flats of Bering River and to the west into the flats of Copper River. It lies, in part, in the southern slope of a densely timbered highland whose summits reach to 1,200 or 2,000 ft. above the sea; in part, in the flats adjacent to the shore line. Drilling has been done at several localities in this belt, but the productive wells are limited to that part of the belt lying between the town of Katalla and Bering River.

The surface rocks consist chiefly of a series of intensely folded and faulted shales, sandstones, and conglomerates, with some small basalt or diabase dikes and sills. The general structural trends are about N.20°E., and the line of seepages lies diagonal to this structure. As much of the field is masked with a dense growth of vegetation, details are lacking on the minor structures. In certain instances, seepages or groups of seepages are closely associated with faults or with subordinate anticlines. It is true, however, that no definite law of occurrence of the petroleum with tectonic features has yet been established. West of Katalla seepages have been found belonging to the same belt, in surface rocks of more or less metamorphosed graywackes and slates.

The shale, sandstone, and conglomerate series is of Tertiary age and probably older than the Tertiary coal measures of the Bering River field which lies about 25 miles to the northeast. All these Tertiary beds are undoubtedly younger than the metamorphosed graywackes and slates, which may be of Mesozoic or of Paleozoic age.

It will be evident from the above discussion that the source of the oil is unknown. A source in the metamorphic graywackes is, of course, very improbable. It may be derived from the Tertiary beds or from strata not exposed at the surface. A possible explanation of the seepages in the metamorphic rocks is that these have been thrust over younger oil-bearing horizons.

Katalla, the distributing point for this field, is a small settlement at which freight can be landed from scows only during favorable conditions of the wind. During the oil excitement some use was made of Controller Bay, 15 miles east of Katalla. Within its shelter ships discharged on scows, and these were landed at the mouth of Bering River. Plans have been formulated for developing the Bering coal field by a branch from the Copper River Railroad, connecting with tidewater at Cordova on Prince William Sound. (See Fig. 1.) Another plan contemplates a railway stretching inland from a terminal on Controller Bay. Either plan could be made to serve the Katalla field with but little additional expense. There is ample timber available for structural purposes.

Yakataga Field

The Yakataga petroleum field lies about 80 miles east of Katalla. Here a series of seepages marks a zone about 20 miles in length and one-half

to two miles from the beach. The extension of this line to the west carries it into the Pacific, and to the east into an unexplored and ice-covered region tributary to Icy Bay. Prospectors report the presence of a strong seepage near Yahtse River, about 15 miles east of the locality to which the belt has been actually traced. There is also a less definite report of the occurrence of seepages along the mountain front between Yakutat and Lituya Bay, about 200 miles east of Yakataga. What little is known of the geology of this region lends some support to this rumor.

The line of Yakataga seepages lies for the most part in a series of short valleys separated from the coastal plain by a low wooded ridge and drained by streams whose courses are transverse to this ridge. About a dozen seepages have been found, most of which are little more than exudations along joint cracks. One, however, on Johnston Creek is roughly estimated to discharge a barrel or more of petroleum a day.

So far as determined, all these seepages lie along a sharp anticline whose southern limb is about vertical and whose northern limb dips inland at from 15° to 45°. The exposed rocks consist of sandstone overlain by fine-textured shale of Oligocene or lower Miocene age. No drilling has been done in this field, so that there is no information at hand of the sub-surface geologic conditions. Speculation as to the source of the petroleum is, therefore, futile.

The Yakataga district is to-day almost inaccessible, as all landings must be made on a beach exposed to the full sweep of the Pacific. Most of the freight for the few placer miners in the district is brought from Katalla by launch when the winds are favorable to a landing. The overland route along the beach is difficult, though it probably could be utilized for a pipe line or narrow-gauge railroad. The marked recession of a glacier at the eastern end of the field during the past few years has revealed a small indentation known as Icy Bay in which shelter may possibly be had, though at present it is in part blocked by ice cakes. If the glacier continues to retreat, the Yakataga field will probably be rendered accessible. There is abundant timber suitable for structural purposes in the district.

Iniskin Bay Field

Iniskin Bay is an indentation about 12 miles deep which, with Chinitna Bay on the north, blocks out an irregular-shaped peninsula on the west shore of Cook Inlet. The more or less even shore line of this peninsula is broken on the southwest by two small indentations—Oil Bay and Dry Bay. Petroleum seepages have been found in this field near Iniskin, Oil, and Dry bays.

The bedrock of the field is a fine-grained sandstone with which are interbedded some clay shales. Some beds of conglomerate occur in the

sandstone, one of which forms the basal member of the formation, and near the head of Iniskin Bay this rests on sheared igneous rocks. The sandstone and the associated sediments, which are of Middle Jurassic age, have a thickness of about 1,100 ft. It is overlain by a shale formation with intercalated conglomerate. The seepages occur in the eastern limb of a broad anticlinal arch which has been somewhat faulted. Drilling has not gone deep enough to indicate whether the altered igneous rocks underlie the whole field as they do at the western margin. The evidence in hand points to the conclusion that the sandstone is the source of the petroleum.

The field is readily accessible from the good harbors lying both north and south. These are occasionally blocked by ice floes, but are usually accessible throughout the year. There is some timber in the district, but this is chiefly of an inferior quality. Timber could, however, be brought from other parts of Cook Inlet region at no great expense.

Cold Bay

Cold Bay is an indentation on the Pacific shore of the Alaska Peninsula nearly opposite the south end of Kodiak Island (Fig. 2). The adjacent area, which is untimbered, consists of rounded hills rising to altitudes of less than 1,000 ft. Above this altitude rise some higher peaks, chiefly made up of volcanic rocks.

The country rock of the area in which the seepages occur is a sandstone and shale formation carrying a little limestone; is of Middle Jurassic age, and about 2,000 ft. in thickness. It is underlain by shales, limestones, and cherts (Triassic) and succeeded by Middle Jurassic arkose conglomerate, sandstone, and shale. The youngest rocks of the district are volcanics, chiefly andesites and basalts. It is a significant fact that the oil-bearing member of this series is of the same age as the oil-bearing horizons of the Iniskin Bay field. The main structural features are broad open folds whose axes parallel the coast, trending about northeast and southwest. The dips of the strata seldom exceed 15°. Some faults have also been noted in the district. There are a number of oil seepages in the field, some of which are strong. At one of these seepages there is considerable flow of gas.

There is a good harbor at Cold Bay open throughout the year, and the contour of the region makes it readily accessible. There is no timber in the district.

Smith Bay Locality

E. de K. Leffingwell, who for a number of years has been exploring the north coast of Alaska, has reported the occurrence of petroleum residue about 100 miles east of Point Barrow. Mr. Leffingwell describes this

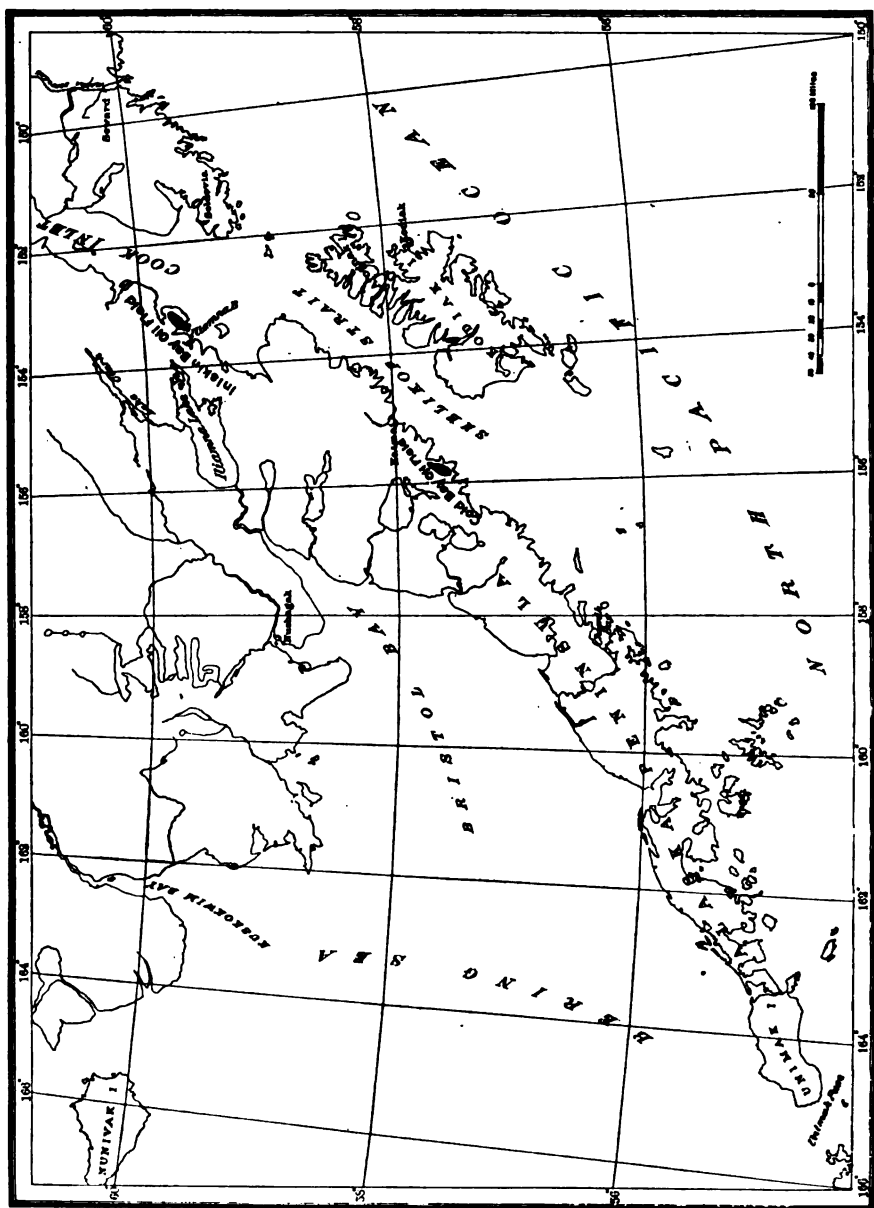


FIG. 2.—MAP SHOWING LOCATION OF INISKIN BAY AND COLD BAY OIL FIELDS, ALASKA.

material as occurring near Smith Bay¹ in a mound several hundred yards in diameter and standing about 150 ft. above the level of the tundra. It contains considerable vegetable matter and silt, but appears to be the residue of a petroleum containing an asphaltic base. Nothing is known of the geology at the locality of the find, but in the general province in which it occurs there is a series of horizontal Tertiary sediments overlying gently folded Mesozoic sediments. Even if an oil pool were found in this northern field the conditions of transportation would prohibit present commercial development. Smith Bay is a shallow indentation and is locked in the ice at least 10 months in the year.

Character of the Petroleum

The Alaskan petroleum, so far as its quality is known, is a refining oil similar to that of Pennsylvania, and has, as already stated, a high percentage of volatile compounds, a paraffin base, and contains but little sulphur. The accompanying table will serve to illustrate the quality of the oils from the Katalla and Yakataga fields, and, so far as known, those from the Iniskin and Cold Bay fields have about the same composition. It should be noted that all the samples from the Yakataga field and part of those from the Katalla field are from seepages where there has been a loss of the volatile compounds.

Development

The presence of petroleum seepages on the Alaska Peninsula seems to have been known to the Russians as early as 1865, but aroused no interest. Oil was reported along the Pacific seaboard in 1880, but it was not until about 1896 and 1897 that definite attention was drawn to the Katalla and Yakataga fields. The first drilling was done at Katalla in 1901. Meanwhile a survey was made for a pipe line from Controller Bay to the Yakataga field, but no drilling has been done in the latter field. In 1901, the first drilling was done in the Iniskin field, and two years later three wells were sunk in Cold Bay.

The enormously rapid increase of the oil output from the California fields caused a sudden collapse of the Alaska oil boom, which reached its zenith in about 1904. Operations in Alaska were very expensive, largely on account of cost of transportation, and, in spite of the high grade of the petroleum found, operators were soon discouraged. Moreover, at the time of the petroleum excitement the local market for the product was inconsiderable. In 1906, all Alaskan petroleum lands were withdrawn from entry, which served as a further deterrent to operations.

¹ Brooks, Alfred H.: The Mining Industry (Alaska) in 1908, *Bulletin No. 379*, U. S. Geological Survey, pp. 61, 62 (1909).

Summary of Analyses and Tests of Katalla and Yakataga Petroleum

Locality	Color	Specific Gravity	Gravity, Degrees Baumé	Flashing Point, Degrees F.	Benzine Per Cent.	Kerosene, Per Cent.	Lubricating Oil, Per Cent.	Residue Coke and Loss, Per Cent.
KATALLA FIELD								
Katalla, well, 10 ^a		0.8280	39.1		21.0	51.0	28.0	
Katalla, well, 10 ^b		0.7958	45.9		38.5	31.0	21.5	9.0
Katalla, well, 10 ^c	Light green.	0.7957	45.9	70-80	38.5	31.0	21.5	9.0
Katalla, well, 10 ^c		0.800			34.2	34.4	16.5	14.5
Katalla, well, 10 ^d	Dark red.	0.802		-60				
Katalla, well, 10 ^d	Dark red.	0.780		-60				
(?) ^d		0.869				19.0	78.6	1.7
(?) ^d		0.914				9.0	87.6	2.7
(?) ^d		0.800			24.8	53.9	16.7	1.2
Katalla Meadow ^d	Dark brown	0.929		240				
Katalla Meadow ^d		0.901		156				
Katalla Meadow ^d		0.874		156				
Katalla Meadow ^d		0.869		152				
Katalla Meadow ^d		0.961		266				
Burle Creek ^d	Dark red-brown.	0.942		234				
YAKATAGA FIELD								
Johnston Creek, ^{d, e}	Dark brown	0.964		200				
Johnston Creek, ^{d, e}	Dark brown	0.879		178				
Poule Creek ^{d, e}	Dark brown	0.970		250				
Poule Creek ^{d, e}	Dark brown	0.881		67				
Poule Creek ^{d, e}	Dark brown	0.914		156				
Crooked Creek ^{d, e}	Dark brown	0.921		172				
Oil Creek ^{d, e}	Dark brown	0.855		108				
Yakogelty ^{d, e}	Dark brown	0.937		246				
Morrison Creek ^{d, e}	Dark brown	0.991		270				
Argyll Creek, Icy Bay ^{d, e}	Dark brown	0.962		310				

Bulletin No. 592, U. S. Geological Survey, p. 146 (1914).

^a Sample collected by G. C. Martin, test by Penniman and Browne. *Bulletin No. 335, U. S. Geological Survey, p. 121 (1908).*

^b Oliphant, F. H.: The Production of Petroleum in 1902, *Mineral Resources, 1902, U. S. Geological Survey, p. 583 (1903).*

^c Stoess, P. C.: The Kayak Coal and Oil Fields of Alaska, *Mining and Scientific Press, vol. lxxvii, p. 65 (1903).*

^d Redwood, Boverton, Petroleum, vol. i, 2d ed., p. 198 (1906).

^e The exact localities of seepages where these samples were taken are not known, but they are believed to be in the Yakataga field.

Assessment work has, however, been kept up on some claims staked previous to the withdrawal. Patent has been granted to a small group of claims in the Katalla field, which constitute the only Alaskan oil lands to which the government has relinquished title.

In all, about 26 holes have been drilled in the Katalla field, of which probably 10 have struck some oil. The deepest well is about 1,600 ft., but the geology is so complex that the actual depth is not significant of the position of an oil pool. Some natural gas was encountered in the drilling. An output of about 2 to 10 bbl. is reported from the productive

wells. In 1912, a small refinery was erected near Katalla. Since then this plant has supplied gasoline, illuminating oil, etc., to various local industries.

In the Iniskin Bay field one well was sunk to about 1,000 ft., encountering gas and oil. A strong flow of water shut off the oil. A second well was abandoned at a depth of 450 ft. A third well met with oil and gas at a depth of 770 ft., and the hole was continued to a depth of 900 ft. There has been no drilling in this field since 1904.

In the Cold Bay field two wells have been drilled to depths of about 1,500 ft., and several oil sands are reported. There have been no developments since 1904.

Summary and Conclusions

The Katalla field is a region of closely folded and faulted Tertiary rocks, possibly comparable in its structure to that of the California fields. Much of the drilling in this field was done without any regard to geologic structures, and but few good logs are available. The presence of oil is indicated by the success achieved, but it will require further drilling to determine the presence or absence of a large pool. At Yakataga the structure of the oil-bearing formation appears to be much simpler, and it ought to be possible to test the field without a very large amount of drilling. }

In both the Iniskin and Cold Bay fields Middle Jurassic sandstone forms the country rock in the oil-bearing area. The open folding should be favorable to oil pools; on the other hand, some faulting has been observed. There is reason to believe that the oil-bearing formation found in these two fields may have a wider distribution in the Alaska Peninsula. Whether it carries petroleum at other localities has not been determined. On the whole, there is good hope of finding petroleum in the Alaska Peninsula.

Alaska affords a steadily growing market for petroleum and petroleum products. In 1913, 15,682,000 gal. of crude oil, 1,735,000 gal. of naphtha, 661,000 gal. of illuminating, and 150,000 gal. of lubricating oil were shipped to Alaska. If any oil field were developed on the Alaska Peninsula, it would be especially well located for supplying the local markets on Seward Peninsula, Yukon River, and Pacific seaboard.

DISCUSSION OF THIS PAPER IS INVITED It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. (Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Improved Methods of Deep Drilling in the Coalinga Oil Field, California

BY M. E. LOMBARDI, SAN FRANCISCO, CAL.

(New York Meeting, February, 1915)

THE Coalinga oil field is located on the west side of the San Joaquin Valley, California. The structure is in general a monocline, the edges of the oil horizon resting on the foot hills and dipping gently toward the east. One prominent anticline occurs plunging southeast. The earlier drilling was done in the foot hills comparatively near the outcrop and the wells were shallow. The sands were followed eastward and, in the case of the anticline, along the plunge, the wells becoming deeper and deeper until the depth of 4,000 ft. was reached and passed. There is nothing to show that the oil will not be found in quantity at still greater depth. In fact, some of the best producers have tapped the sand at close to 4,000 ft. The recovery of oil still farther to the east, and therefore at greater depth, seems to be mainly a question of drilling.

In this territory the formations drilled through are chiefly sands and shales; they will not "stand up" in an open drilling hole; the casing has to be carried close to the bit, and it is always difficult to keep the casing free for any considerable distance.

Ability to carry casing of comparatively large diameter without conductor pipes for distances of 2,000 or 3,000 ft. or over is desirable in such territory chiefly for two reasons. It makes it possible to enter the oil sand with a pipe of ample diameter; it eliminates one or more expensive strings of casing which act only as conductors for the water string, and furthermore, in territory where waters are encountered which corrode steel rapidly, it makes possible the construction of a rust- and alkali-resisting water string.

It is always desirable to shut off top waters, which may lie within 100 ft. or less of the oil sand, with 10-in. pipe. Where the depth is so great that a practical weight of 10-in. pipe will not withstand the probable collapsing pressures, $8\frac{1}{4}$ in. at least is desirable.

About the limit of rotary drilling to date in California seems to be the setting of the 10-in. string at 3,200 ft., although the rapid advance in rotary work during the past year seems to indicate that this depth may soon be increased. It is my purpose now, however, to treat only of cable-tool drilling.

The problem is to reach a depth of 4,000 ft. or more with a string of pipe not less than $8\frac{1}{4}$ in. in diameter for shutting off top water, and to reach it with this string free and movable and using in the upper part of the hole the minimum of conductor casings.

In the Coalinga field some very promising results have recently been obtained by a method, or combination of methods, effected by William Keck. In one well a $15\frac{1}{2}$ -in. string was set at 2,300 ft., and a $12\frac{1}{2}$ -in. string through this at 3,003 ft., these being the only strings used in the well to that depth and both being landed when they were entirely free. In another well a $15\frac{1}{2}$ -in. string was set at 2,100 ft. and through this a 10-in. string at 3,300 ft.; in this case also the strings were perfectly free when landed and were stopped only because it was not desired to carry them deeper. Other wells have shown similar results.

It will be noted that the $15\frac{1}{2}$ -in. strings were free with over 2,000 ft. of "friction" on them, and this in a territory that will not "stand up" with ordinary drilling more than 40 or 50 ft. ahead of the pipe.

The drilling detail used on these wells is briefly as follows: A large clearance for the pipe is obtained; the standard circulator system is used; the pipe is kept moving while drilling is in progress; each collar is "set up" twice before it goes below the bottom of the derrick cellar.

The large clearance is obtained by the use of a shoe of extra large diameter, from $1\frac{1}{4}$ to $1\frac{3}{4}$ in. larger than the collars on the pipe. Under-reaming is resorted to frequently and the hole is repeatedly under-reamed until the pipe is entirely free in passing a "shell" or hard streak. Spudding the pipe is avoided. When a conductor pipe is landed the desirable extra clearance for the next string may be obtained by skipping one size of pipe, as for instance carrying a 10-in. string through a $15\frac{1}{2}$ in., thus eliminating the $12\frac{1}{2}$ -in. size. This is necessary with the present sizes of pipe available, but a different design, which will be mentioned later, would save considerable expense in this matter.

This extra clearance is necessary in using the circulator system to allow free passage for the "returns." It obviates the danger of sand lodging between the strings of pipe and freezing the working string.

The well-known standard circulator system is used. Mud- (clay) laden fluid is forced down through the pipe under pressure by pumps (ordinary rotary slush pumps) and returned on the outside of the pipe, carrying the drillings with it. This fluid is run through a flume and into a pit, as in rotary work, and its consistency is regulated as with the rotary.

This mud-laden fluid presumably plasters up the walls of the hole, prevents sand and mud from running in and prevents caving. It is essential that circulation be interrupted as little as possible. Intermitent circulation seems to be worse than useless.

The pipe is kept moving while drilling is in progress—i.e., without pulling out the tools—by means of a so-called "swinging spider" (see

Figs. 1 and 2). The pipe is suspended by an ordinary spider provided with lugs to which are attached steel reins (sometimes chains or wire lines) which extend to a clevis above the walking beam, the beam operating between the reins. The clevis is attached to the casing block. The reins are about 40 ft. in length so that the pipe may be lowered to the bottom of a 30-ft. cellar. The cellar is made deep enough so that the stationary spider at the bottom is more than the length of one joint of pipe below the derrick floor. It follows that when a joint of pipe is added to the string the back-up tongs may be put on the second collar, which has been previously set up and which is now near the cellar bottom. The

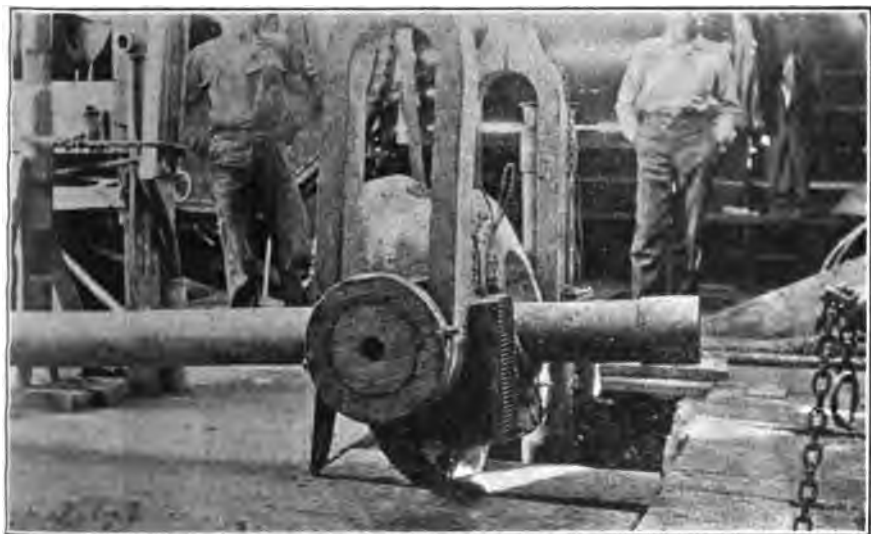


FIG. 1.—SWINGING SPIDER

same result is obtained without back-up tongs, the pipe being held by the lower spider. Thus every joint is set up twice—once when it is put on and a second time after it has been subjected to the pull of the pipe below it and the vibration of drilling. This insures a tight joint.

It is by a combination of the above details and careful attention to them that success in carrying pipe has been attained.

Other advantages are obtained, one might say, as by-products of this method. The pipe is always free and the circulation perfect for cementing, the mud being easily washed out ahead of clean water. There is almost total elimination of bailing out drillings, with its consequent loss of time. A lifting pressure may be put against the closed top of the casing, thus relieving to some degree the strain on the casing line. For instance, a 12½-in. casing has an area of about 121 sq. in.; a pumping pressure of 200 lb. per inch against this means 24,200 lb. taken off the

effective weight of the casing. Naturally the pressure runs up if the pipe becomes logy and that is when it is most needed.

The mud-laden fluid as usually used has a specific gravity of about 1.40; therefore, its pressure in holding back artesian water, running sand,



FIG. 2.—TEMPER SCREW WITH CLAMPS ON DRILLING LINE BETWEEN REINS OF DRILLING SPIDER.

etc., is 1.4 times as much as clear water. It is a well-known fact that this mud-laden fluid tends to kill gas (see *Technical Paper No. 66, U. S. Bureau of Mines, Mud-laden Fluid Applied to Well Drilling*), although it is the

writer's opinion that the capillary action of water in sand has as much to do with holding gas pressure back as anything.

It has been suggested that in territory where upper waters corrode iron and steel very fast, a 10-in. water string practically immune to this corrosion may be obtained as follows:

Carry a 12½-in. string within a few feet of the point where water is to be shut off. This string may be as light and cheap as it is practical to carry, since the burden of sustaining the collapsing pressure of the water does not fall on it. Then land a 10-in. water string inside of this 12½-in. string at the proper point below it, and pump in enough cement to fill the space between the two strings.

This is mentioned simply as one of the advantages which may accrue from a drilling method by which a large-diameter string of pipe can be carried to depth with reasonable certainty.

Now as to the interesting item of cost. The bulk of the extra cost incurred is in movable tools and machinery, only the depreciation and upkeep on which are chargeable to the well drilled. Extra cost incidental to the drilling itself, other than above, consists in construction of the deep cellar, the installation of one extra boiler, the mud pumps and the extra fuel and water used, and one extra man. The swinging spider, pumps, boiler, etc., are, of course, moved and used for successive wells.

An idea of the extra cost items may be gained from the following:

Extra depth of cellar, drain, and circulator rigging.....		\$450.00
Setting of extra boiler and circulator pump.....		\$250.00
Mud flume, pits, etc.....		\$100.00
Total fixed costs per well.....		\$800.00
Extra labor in drilling, per day.....	\$7.00	
Extra fuel, 7 bbl. of oil per day at 35c.....	2.45	
Extra water, packing, etc., per day.....	4.00	
Total extra costs per day for 122 days.....	\$13.45	1,640.90
Depreciation at 18 per cent. on \$2,320, being value of outfit removed.		136.00
Total extra cost.....		\$2,576.90

In a typical well with the system under discussion 3,336 ft. of 10-in. pipe was set in 122 days.

This is the value of only about 1,465 ft. of 12½-in. 45-lb. casing f.o.b. the field. No 12½-in. casing was used, so at least this amount was saved.

Better average time is made with this method, so that at the most it is not more costly than the usual cable-tool drilling. As pointed out, its chief value lies in the fact that large-diameter pipe can be carried to depth with far more certainty.

It is impossible to leave this subject without a few suggestions for

the future; in other words, indulging in a mental construction of an ideal drilling outfit, built along lines following the above described method.

A greater clearance between consecutive size strings of casing is essential. In order to shut off water with a certain size string, the conductor used, be it long or short, should have an inside diameter at least $1\frac{3}{4}$ -in. greater than the outside diameter of the collars on the water string. Since the water string will have to stand great collapsing pres-

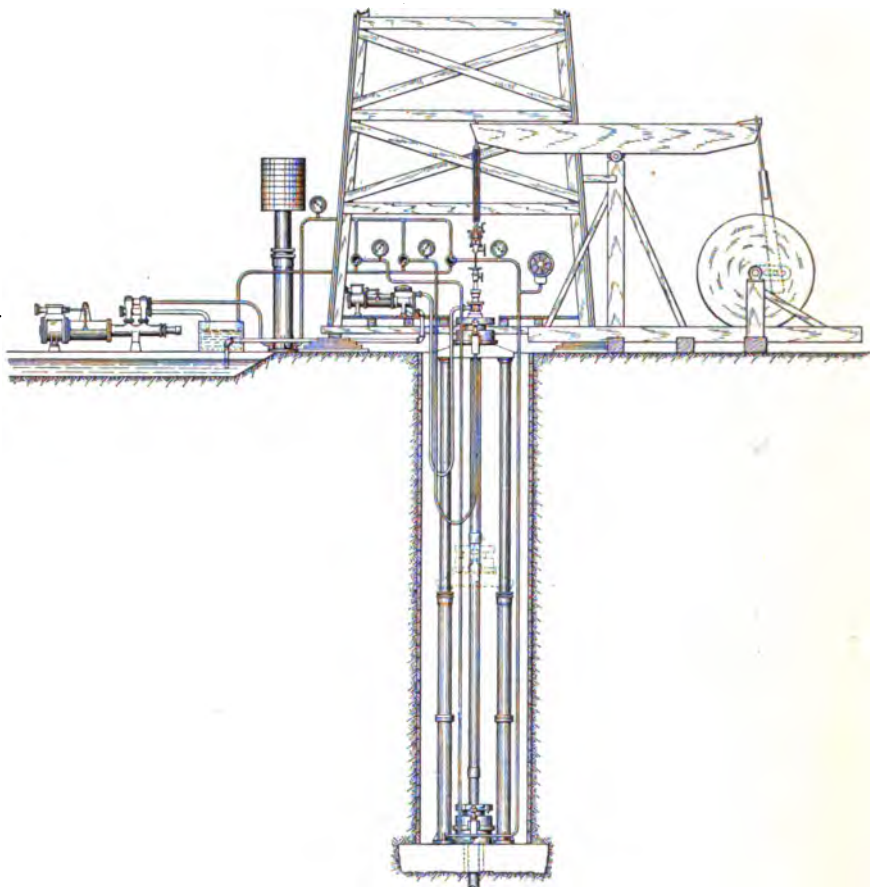


FIG. 3.—HYDRAULIC ELEVATOR FOR MOVING CASING, USED IN CONNECTION WITH CIRCULATOR SYSTEM. (PATENTED.)

sure and must therefore have very thick walls, a clearance should be provided in it so that the oil string may be worked without difficulty.

If it is desired to use an $8\frac{1}{4}$ -in. oil string, it follows that the water string should be at least $10\frac{1}{8}$ -in., inside diameter, which would be obtained in a casing of $10\frac{5}{8}$ -in. nominal diameter weighing from 55 to 60 lb. per foot. This would call for a 14-in. conductor.

Requirements for the ideal casing are:

Sufficient thickness of walls to withstand the water pressure; joints that will hold on a maximum pull; threads that will stand being set up several times; reduction of weight near the top of the string.

To meet these requirements the casing will have to be heavy on the bottom and light on top, but the top part will have to stand the greatest pull. Obviously an upset-end casing, upset on the outside, would be the thing for the top part of the string. Furthermore, with these upsets and with the thickness of the wall necessary on the bottom of the string, eight threads per inch could safely be used in the joints. It is well known that eight threads will stand more unscrewing and screwing up, more driving and, in general, more "grief" than the customary 10 threads.

Collars may be built correspondingly heavy since excess clearance is obtained anyhow. In explanation of the desirability of using eight threads I would say that in carrying a string of casing a long distance there are generally accidents, such as pinching a shoe, etc., which necessitate pulling and putting back the casing before it is finally landed.

The swinging spider, although efficient, is clumsy. In moving casing with it in the ordinary way, the entire strain (sometimes 80 tons or more of casing, plus friction of the mud between the casing and the walls of the hole, must be moved) is transmitted to the crown block on top of the derrick. To obviate this and carry the greatest strains on solid foundation, a hydraulic elevator operating in the cellar might be used. Such an elevator with a lift of 22 ft. has been devised, but never used to my knowledge. (See Fig. 3.) By its use pipe could be kept moving all the time, and all hard pulls taken off the crown block. Of course, the ordinary elevators and calf wheel would still have to be used for putting in and pulling strings of pipe, but the heavy work could be carried by the hydraulic elevator in the cellar, and the cost of constant repairs to the derrick considerably lightened. Pulling in of derricks, with consequent delays, frozen pipe, and danger to life, now far too frequent, would be almost entirely done away with. With the above suggested improvements, it seems reasonable to expect that an $8\frac{1}{4}$, 10, or $10\frac{5}{8}$ -in. water string, with only a few hundred feet of conductor, could be carried in the territory under discussion to 4,000 ft. or more.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Comparative Costs of Rotary and Standard Drilling

BY M. L. REQUA, SAN FRANCISCO, CAL.

(New York Meeting, February, 1915)

In the fall of 1910, the Nevada Petroleum Co., operating in the Coal-inga field in California, determined to drill a number of wells with rotary tools, in order to prove conclusively the relative value of the rotary as compared with the standard rig.

At that time, the rotary was but little known in California and its proposed introduction met with considerable criticism on the part of the operators and quiet opposition among the standard drilling crews. There were few men available who were competent to handle a rotary outfit, and these few had had but little experience in California fields. Machines as then built were much inferior to those now used, being lighter in construction and of a poorer quality of material. The shells encountered at depths between 1,700 and 2,000 ft. seemed to be too hard to permit of successful drilling, and at these depths the rig was changed over to standard tools and by them completed.

Because of the heavy depreciation, the time lost in converting from rotary to standard, and the comparatively small profit, it was concluded that unless in the future there was some material improvement in rotary rigs, nothing would be gained by drilling with rotary tools upon this property. Little or no drilling has been done upon the property since that time, but in the meantime a large number of men have been trained in the use of the rotary; in fact, many standard-tool men have abandoned standard drilling and, starting in as roustabouts, have become thoroughly competent rotary operators. The improvements in the machinery have been such as to remove many of the objections and the rotary drilling of to-day is in every way superior to that of 1910.

In contrast, a well drilled recently by the Kern Trading & Oil Co., in the east side Coal-inga field may be cited. The first 2,500 ft. was drilled in 24 days and it is confidently expected that the water string will be set with a rotary at 3,400 ft.—a record obviously far beyond that made in 1910 by the Nevada Petroleum Co.

The records herewith are published with the hope that some one having more recent data will publish them so that comparison may be

TABLE I.—*Comparison of Cost of Drilling by Rotary and Standard Methods*

NOTE.—Common costs (as derrick, engines, boilers) not included.

	Rotary										
	2A 30	3A 30	5A 30	6A 30	7A 30	1A 80	8B 30	4 18	5 18	5A 18	Average
	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$	\$
Setting up rotary.....	0.143	0.183	0.217	0.194	0.141	0.135	0.186	0.160	0.212	0.292	0.186
Tearing down rotary.....	0.023	0.047	0.023	0.026	0.049	0.054	0.046	0.027	0.053	0.029	0.037
Stores to rotary.....	0.224	0.280	0.397	0.374	0.325	0.343	0.203	0.095	0.177	0.333	0.270
Extra materials on rotary.....	0.109	0.096	0.095	0.108	0.100	0.111	0.095	0.109	0.108	0.119	0.105
Extra labor on rotary.....	0.024	0.021	0.021	0.023	0.022	0.024	0.020	0.022	0.023	0.025	0.022
Fuel oil.....	0.377	0.221	0.193	0.237	0.295	0.150	0.256	0.194	0.305	0.235	0.241
Water.....	0.029		0.047	0.029		0.046	0.037	0.070			0.037
Drilling labor.....	1.316	0.869	1.000	1.027	1.367	1.156	1.192	1.395	1.376	1.725	1.242
Standard crew casing.....	0.197	0.210	0.531	0.378	0.184	0.336	0.250	0.283	0.254	0.087	0.271
Rig depreciation.....	0.298	0.266	0.260	0.294	0.377	0.302	0.259	0.331	0.295	0.326	0.300
Casing depreciation.....	0.274	0.270	0.220	0.276	0.274	0.247	0.218	0.219	0.108	0.218	0.232
Overhead charges.....	0.242	0.102	0.079	0.211	0.205	0.094	0.170	0.081	0.157	0.194	0.153
Depreciation on tool joints, etc.	0.149	0.134	0.131	0.149	0.139	0.153	0.130	0.151	0.150	0.164	0.145
Extra pump connections.....	0.013	0.012	0.016	0.014	0.011	0.014	0.010	0.013	0.012	0.014	0.012
	3.418	2.658	3.231	3.240	3.389	3.165	3.084	3.122	3.242	3.766	3.241
Depth, feet.....	1,880	2,123	2,150	1,902	2,036	1,846	2,151	1,872	1,884	1,711	1,955.5
No. of days drilling.....	55	62	55	56	68	54	62	62	63	65	60.3
Av. no. ft. drilled per day.....	32.2	34.2	39.1	33.9	29.9	34.2	34.6	30.2	29.9	26.3	32.46

Standard (1955 ft.)

	4A-30	8A-30	4A-18	Average
	\$	\$	\$	\$
Materials used.....	0.637	0.651	0.623	0.637
Labor.....	1.127	1.355	1.378	1.287
Overhead charges.....	0.160	0.101	0.136	0.132
Oil.....	0.235	0.296	0.417	0.316
Water.....	0.018	0.060	0.072	0.050
Depreciation charges.....	0.995	0.995	0.995	0.995
Cost per linear foot.....	3.171	3.458	3.612	3.417
Total days drilling.....	71	95	99	88.3
Depreciation charges made as follows:				
Bull band, calf wheel.....	247.50		($\frac{1}{2}$)	82.51
Steel crown block pulley.....	258.75		($\frac{1}{2}$)	86.25
Excess casing.....	2,561.08		60%	1,024.40
Walking beam.....	35.00		10%	3.50
Drill tools.....	3,483.75		20%	696.75
Sand line.....	182.00		25%	45.50
Cokely truss rod.....	35.00		20%	7.00
Total depreciation.....				1,943.00
Depreciation per foot.....				0.995

SUMMARY

Average standard time	= 88.3 days	Production assumed at 250	
Average rotary time	= 60.3 days	bbl. per day, 28 days	= 5,600 bbl.
Favor rotary	= 28 days	Favor rotary	
Standard average cost per ft.	= \$3.42	Oil 5,600 bbl. at 50c.	= \$2,800.00
Rotary average cost per ft.	= 3.24	Drilling 1,955 ft. at 18c.	= 352.00
Favor rotary	= 0.18	Total	= \$3,152.00

made between 1910 and 1914. The wells drilled comprise 10 in number. The property is located about 2 miles from the railroad track and on practically level land, so that costs of hauling are a minimum. Costs would be higher upon a property at all remote from transportation, or where the ground was hilly, or the lease was not fully equipped as a going concern.

In order to compare intelligently the work done, careful costs were kept and compared with the costs of standard drilling as previously practiced upon the same property. Table I gives the detailed costs of drilling the rotary and standard wells and comparative summary. Table II shows the advance of each well for various periods of time from the commencement to the completion.

TABLE II.—*Depth of Wells on Various Days*

Days	2A 30	3A 30	5A 30	6A 30	7A 30	1A 30	8B 30	4 18	5A 18	5 18
10	90									100
15	540	500	100		50	400	100	170	150	450
20	800	780	400	70	200	800	300	300	400	800
25	1,050	1,150	900	350	500	1,100	600	650	670	1,050
30	1,300	1,380	1,300	750	800	1,340	800	880	950	1,280
35	1,440	1,550	1,500	1,100	1,050	1,500	1,000	1,150	1,070	1,450
40	1,550	1,700	1,680	1,450	1,280	1,700	1,280	1,400	1,180	1,580
45	1,680	1,850	1,900	1,650	1,420	1,810	1,550	1,600	1,330	1,660
50	1,780	2,010	2,050	1,780	1,590	1,830	1,750	1,690	1,530	1,710
54						1,846				
55	1,880		2,150		1,700		1,950	1,760	1,650	1,800
56				1,902						
60					1,850					
62		2,123					2,161	1,872		
63					2,036					1,884
65									1,711	

Rigging up time as follows is included in above figures:

2A, 7 days

6A, 14 days

8B, 7 days

3A, 11 days

7A, 8 days

4-18, 10 days

5A, 12 days

1A, 11 days

5-18, 10 days

5A-18, 8 days

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The Rôle and Fate of the Connate Water in Oil and Gas Sands

BY ROSWELL H. JOHNSON, PITTSBURGH, PA.

(New York Meeting, February, 1915)

WHAT becomes of the water which must have filled the oil and gas sands at the time of deposition, has long puzzled students of oil and gas and has found expression in Munn's well-known article on the hydraulic theory. Both Munn and Clapp have pointed out that in the Appalachian field the deeper sands carry increasingly less proportionate amounts of water.

It seems to me probable that the principal source of oil and gas has been in the shales in proximity to the sands which now contain these products. The sand spits which mark off the lagoons from the sea comply particularly with the requirements of a limited sand imbedded in shales rich in organic matter.

Cunningham Craig has objected to this view that the rich organic muds of lagoons are found only at the surface. That such muds do extend to considerable depth I am assured by Prof. Douglas W. Johnson, who has made an extensive study of the Atlantic Coast marshes.

As any particular sand body becomes weighted by a heavier and heavier overburden, the result of increasing deposition, a part of the very large percentage of water in the uncompacted sand and mud is forced out. The water percentage of freshly settled mud is vastly higher than that of the resultant shales, or even of a very porous sand. If beds of less compressible material meet or underlie the shales in question, there will frequently be a lateral motion along such beds to some point where the upward movement will be resumed.

We find in our laboratory work here that oil, gas, and water have extremely slight capacity for gravitational sorting while in a state of rest but when moving the gravitational sorting is readily accomplished. As the oil, gas, and water pass a body of larger pores, the gas, owing to its lack of capillary attraction, is retained in the large pores.

This has been predicted by Washburne, Blatchley and others on the ground that the water has a higher capillarity and grips the finer pores in such a way that there will be a greater proportion of oil and gas in the large

pores. Washburne calls this "capillary concentration." I would prefer the term "selective segregation," because it seems quite possible that other factors contribute as well as differential capillarity. The greater viscosity of oil would cause it to move more readily in larger channels than in small, so that it would with greater difficulty leave the large channels for the smaller ones than would water. Again, even if the viscosity were the same, the walls of the pores are originally water-wet, so that any immiscible fluid such as oil would find penetration into fine pores more difficult than would water and there would be a consequent selective segregation in the larger pores. A fourth factor is the circumstance that the selective segregation of gas in the larger pores would hold some oil associated with it as a pellicle, once oil touched the gas surface, as I have previously pointed out.¹

We have actually demonstrated in sand arrangements in percolators that the percentage of Texas oil in a coarse-grained body of sand surrounded by a very fine sand becomes excessive where a mixture of oil and water is driven through a water-wet sand. Details of this experiment will be given. A series of experiments is now in progress to distinguish the relative effectiveness of these factors.

The original contents of connate water, its motion caused by compacting, and the selective segregation of oil and gas in the large pores, make it possible to believe that the oil and gas arise in the shale and become segregated in part in the sand, but with much oil and gas remaining in the shale.

David White finds it very desirable to divide the processes of coal formation into two stages, the biochemical and the dynamochemical. He recognizes that methane was formed in the first by the action of bacteria and in the second by the action of pressure. Evidence will be presented to show that we have two periods of natural-gas formation from the organic matter in shales. The first, while the deposit is still shallow and the formation unconsolidated, is effected by the action of anaerobic bacteria. After this action has ceased, a second period of gas formation begins as a result of pressure upon organic matter in the shale. This continues for a very long period. The ascent in lesser quantities of abysmal gas, in part inorganic in origin, as is indicated by the frequent presence of nitrogen, argon, and helium in the deeper sands of Kansas, as shown by Cady and McFarland, contributes also to the high pressure.

A further cause of movement and probably increased pressure with depth, and therefore consequent gas formation (if the hypothesis be correct), is the gradual deposition of cementing material in the sandstone. This reduces the percentage of pore space and displaces material previously occupying such space. The water (with oil and gas, if any) moves

¹ *Science*, new ser., vol. xxxv, No. 899, pp. 458 to 459 (Mar. 22, 1912).

in the direction of least pressure, which is generally upward, as before. But with lowered pressure the water deposits more cementing material, as its solvent ability is thus reduced. The ultimate source of the cement contributed to sandstone is then principally from below. The paths that water takes in its upward course become, by virtue of this fact, clogged, and the difficulties of its circulation are increased with a resulting increase of the pressure gradient.

It has been common to explain the pressure gradient by the "head" of water column constituted by the intercommunicating channels, but the "string of bead"-like arrangements and the tortuosity and fineness of the pores may well make us incredulous of the efficiency of such a head, until experimental demonstration is forthcoming. "Rock pressure" from mere weight of overburden, while generally felt to be inadmissible for moderate depths, can be appealed to at great depths, as well as the expansion with rising temperature of material contained in the pore spaces, as the accumulation of overburden raises the isogeotherms.

We find, then, that the oil and gas reservoirs, when newly compacted, contain a large percentage of the oil which they are to contain. From this time on, there is a gradual formation of cement in the sandstone, which involves a reduction of its percentage of pore space. There is thus forced out of the reservoir some more of its contents; but it will be the water that will be driven out, for reasons given above. The gas which is associated with the oil is gas of the heavy type, such as is usually formed with the oil; but now, owing to the increasing pressure of the continually heavier overburden, we have an additional source of gas formation, namely, the effect of pressure on organic material other than oil which may be in the rock. If we may believe the positive statement of Cunningham Craig, in *Oil Finding*, experiment has actually produced methane in this way. Such a gas would be presumably a "dry" gas.

This recently formed gas, being produced in excess at the points of abundance of the materials from which it was formed, will move through communicating channels in various directions, and will be in large part selectively segregated in the sands. But in so doing it will be a further cause of the displacement of water from these reservoirs.

The expelled water will meet a greater resistance below, owing to the fact that similar processes (only more efficient, because of the greater pressure) will prevent movement in that direction. The amount of possible lateral movement is too limited to offer any important relief. We are forced to believe, therefore, that the expelled water moves, in general, upward, although lenticularity of beds, unconformities, etc., will complicate the course while leaving it essentially a rising one.

Does this not explain why we have, in general, increased gas pressures with increased depth, in view of the fact that direct rock compression and

hydrostatic pressure have been abandoned, for good reasons, as explanations?

Does the adoption of these hypotheses have any bearing on the practice of oil prospecting? It seems to me that, if they are correct, the following practical conclusions result: First, gas is not to be expected in large quantities and under high pressures in relatively recent unconsolidated deposits. Secondly, oil may be expected in good quantities in suitable sands lying in oil shales, no matter how recent, provided only that there has been a proper amount of compacting. Thirdly, a columnar section showing more shale than sandstone, and with its sandstone separated into several beds, rather than constituting one massive bed, may be expected to make available a larger proportion of the oil which was originally formed. If, on the contrary, the view of I. C. White, Thompson, *et al.*, that oil is indigenous to the sands where it is found be accepted, then the ideal section would be one predominantly of sandstone with merely enough shale to act as partitions in facilitating accumulations.

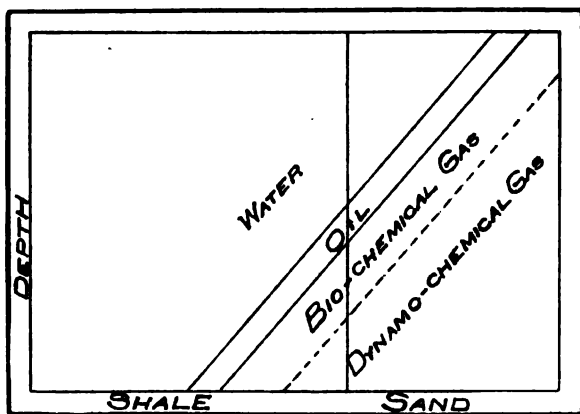


FIG. 1.—DIAGRAM SHOWING RELATIVE POSITIONS OF WATER, OIL, AND GAS.

Fourthly, uniform massive limestones have been rendered so compact by both pressure and crystallization that the oil and gas have been expelled therefrom except in quantities too small to be commercially important. It is highly probable that any porous stratum interbedded with the limestone could receive the oil as it was expelled from the compacting limestone. If we had porous beds more frequently in limestone, we might expect them to be much more important oil-bearing horizons. This view is borne out by the oil formed in the cavities in corals in the Onondaga limestone at Williamsville, N. Y., and yet the absence of oil from the cavities of fossils in many other limestones indicates that some favorable condition is lacking. Is it not possible that in these cases the depth was too great for the necessary bacteria?

With increasing depth, we may expect the reservoirs to show a decreasing proportion of water and increasing proportions of oil and gas, but with the oil not decreasing till a moderate depth is passed, as in Fig. 1; after which there is a decreasing proportion of oil. This assumes that the section continues favorable in essential features for oil and gas. These will vary to such a degree as to make exceptions to the foregoing rule. The gas man may therefore look with more reliance on deep reserves than can the oil man. In the very deep sedimentary horizons, the oil is not absent, if this hypothesis be correct; but is unavailable because, the water having been displaced, the oil has been also in turn displaced from the reservoirs (which are sufficiently porous to be yielding) into the tight grip of the shales.

The freshness of the water in the shallow beds does not make this view untenable because (a) relatively little of the deeper waters reach the surface, owing to the interposed barriers; and (b) these shallow waters are so much freer in their motion that the slow contributions from below produce little effect upon them.

The view of the ascent of the gases and fluids resulting from a strong pressure gradient from the causes given, rather than a descent of shallow waters to replace a mysteriously lost connate water, has a further application in a consideration of the Gulf Coast oil fields.

It has been supposed that oil was lost at an outcrop because water displaced it by gravity. The theory of ascent leads us to the conclusion that it appears in part in some cases as a result of pressure from below. Fortunately the alternation of coarse and fine rocks through which it passes holds back a good share of the oil. In the Gulf Coast we have an excellent example in the salt-oil-sulphur-gypsum domes. At these vertical passageways we have had not a descent but an ascent, with a consequent formation of the dome. In the Mexican fields the deposit of these substances along intrusives, as well as at faults, are due to the passageways opened by the intrusion, jointing of the intrusive, and attendant disturbance of the adjacent strata, to the ascending current with its load.

CONCLUSIONS

It follows from the position here taken that, aside from deviations caused by variations in the favorableness of the accompanying source of hydrocarbons and other modifying influences, we should find an increase of gas with depth, and decreased quantities of oil, as it becomes displaced by the increasing amount of gas, finding its way into the large pores. This is shown diagrammatically in Fig. 1. Should this hypothesis prove correct, the following practical results follow:

1. Much of the world's oil is lost to recovery and there is less hope of deep reserves than we have believed.

2. There are enormous reserves of deep gas, not only in present producing areas but in many others, because, in the absence of water, many more areas without favorable structure will be productive of gas.

3. As the pumps remove the dry gas in these deep pools, the oil back in the shales will release some of its vapors; so that some dry gases will later on, under vacuum, become wet gases and may be profitably compressed, so that the lost oil will not be wholly unrecoverable.

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Housing and Sanitation at Mineville

BY S. LEFEVRE, MINEVILLE, N. Y.

(New York Meeting, February, 1915)

THE solution of the housing and sanitation problem in mining communities, keeping in view both economic and humanitarian aspects, demands the best thought of the management of such enterprises. Upon the solution depend the health and comfort of the employee and his family, and as a consequence his contentment and efficiency.

The following description covers briefly the methods installed for Witherbee, Sherman & Co. The operation of the system has been for about two years under the direct supervision of T. A. Hammond, Land Agent, and I am indebted to him for a compilation of the results.

Sanitation

The sanitary service covers 238 tenements. The installation cost was about \$22 per tenement, as follows: Privy, \$15; privy box, \$3; proportion of incinerator cost, \$2; garbage can, \$2. The operating cost is \$1 per month for each tenement.

In considering the installation of an ordinary village system of water supply and sewerage two difficulties were encountered:

First, lack of a sufficient quantity of water; there is no stream large enough, as Mineville is situated very close to the divide between the Lake Champlain and Hudson River watersheds. A storage reservoir was tried, but the only location available, among glacial hills, would not hold water.

Second, the expense of digging ditches for the distribution of water and sewer pipes is so great as to be prohibitive; Mineville is built chiefly in a valley covered in the bottom with drift, in this case a combination of boulders and hard pan. Digging in this material costs 80c. to \$1 per cubic yard, and pipes must be placed 4½ ft. down to be below frost. The houses are scattered along 4 miles of streets, the oldest having been built in the original forest 65 years ago. The newer houses have gone up a few at a time wherever a clear space could be found and the slopes were not too steep. This has advantages, as it separates the dwellings into

various groups, which makes it possible to segregate the different nationalities; thus we have an American quarter, an Italian, and a Polish-Slavish-Hungarian district.

A water pipe leads to the works for boiler purposes and this supplies the houses along its line. One group of six obtains its water from a well on the hill above; a windmill pumps into an adjacent concrete tank. In

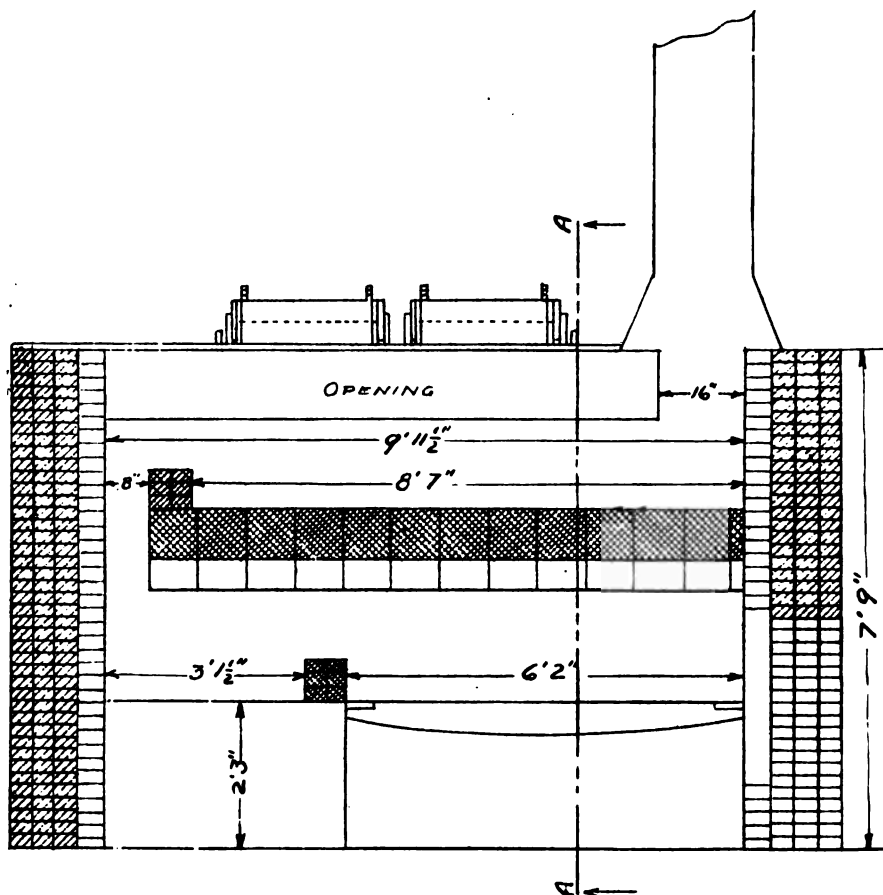


FIG. 1.—LONGITUDINAL SECTION OF INCINERATOR.

several other cases an electric pump in the cellar delivers the water from a well into an air-pressure tank, also in the cellar. Sixty per cent. of the houses are supplied from wells. To keep them free from contamination the old vaults have been abolished, and each privy supplied with a 16-gauge galvanized-steel box, size 14 by 14 by 30 in. Three hundred of these boxes were made to our specifications by the Lyon Metallic Manufacturing Co. They are soldered and riveted water-tight, have

one drop-handle on each end, and cost \$2.73 delivered. With them were supplied 24 covers with handles and with rims 2 in. deep, the price being \$1.25 each. The covers are kept on the boxes only while in transit from privy to incinerator. They are fitted with a strip of rubber packing on the inside next to the rim, and clamps are used to press the cover against the upper edge of the box, preventing the contents from slopping out. After the boxes have been emptied into the incinerator they are taken to a washing rack, thoroughly cleaned and set aside to dry. Before going into service again each box has a small quantity of lime placed in it. The small privies are built with one seat, the larger ones having partitions

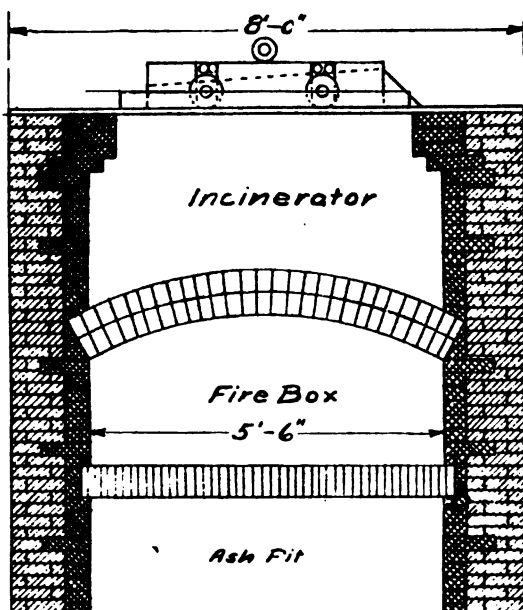


FIG. 2.—SECTION ON LINE A-A OF FIG. 1.

built between seats. The average cost is \$15 set up. They are furnished with door and window, the latter being kept screened. The metal box is placed close up under the bottom of the seat, sliding in from the rear or end through an opening which is kept closed by a tight-fitting door. The boxes are removed frequently and are not allowed to become full. By this system we have been very successful in preventing the contamination of wells and springs.

A garbage can is provided at each house. These are emptied once a week, or oftener if necessary. They are No. 1 Witt cans with tight-fitting lids, made by the Witt Cornice Co., size $15\frac{3}{4}$ by 25 in., capacity 20 gal., weight 26 lb., made from one piece of 23-gauge, 2-in. corrugated galvanized steel, braced with heavy steel bands at top and bottom, and

cost \$1.93 each at Mineville. They have kept their shape, stand handling and are satisfactory.

Incinerator.—An incinerator, shown in Figs. 1, 2, and 3, was built at a cost of \$500, having a firebox of boiler grates 5 ft. 6 in. wide by 6 ft. 2 in. long. A brick arch over the firebox also makes the bottom of the incinerator chamber. An opening behind the bridge wall allows the flame to go above the material to be consumed and up the stack at the fire-door end. This consumes all the gases. No smoke is seen after the fire is started and there is not much odor. The location is about one-eighth mile from the nearest houses.

It was thought when built that the capacity would be large enough

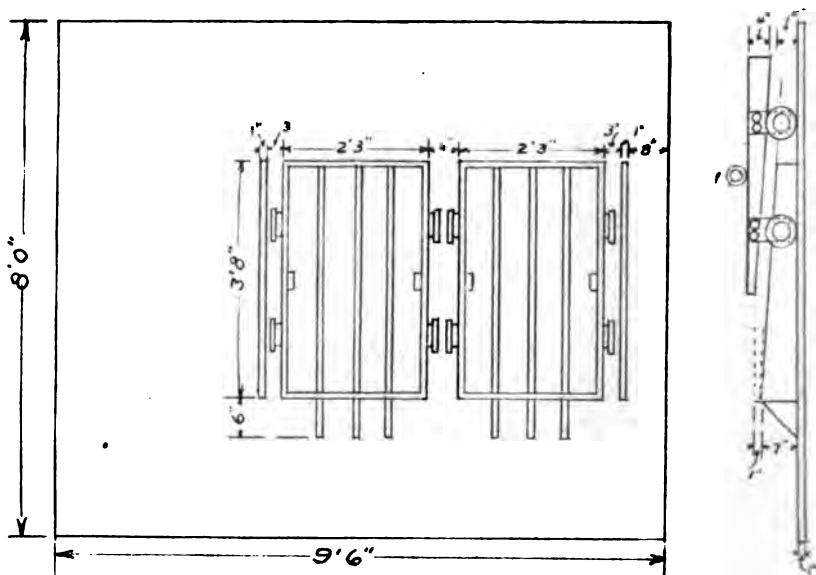


FIG. 3.—TOP VIEW OF INCINERATOR, SHOWING CONSTRUCTION OF TIGHT-FITTING DOORS.

to burn the material from garbage cans also, but it has been found that the incinerator will only take care of the privy boxes. A charge is put in each morning, after which the fire is started and kept going all day, being allowed to go out at night. In the morning the incinerator is cold enough to permit the cleaning out of the ashes, when needed, or for another charge. The top is cast iron, with two feed openings 3 ft. 8 in. by 2 ft. 3 in., closed by cast-iron doors on wheels, having a sloping bottom which comes in contact with a similar slope on the sides of the feed opening in the iron top. This contact makes a tight joint, yet a slight lift with a bar frees the door and it rolls open easily on the wheels.

The material from the garbage cans is now burned in heaps on the ground in the open air and is disposed of in this way without any trouble.

When the incinerator is rebuilt a double grate will probably be provided. The garbage, kept from packing too tightly for the gases to pass through by baffle rods, would be placed on the upper grate, where it would dry and afterward burn. In this way some fuel value would be secured from the garbage.

The cost of operating for the first six months of 1914 was: Labor, \$1,200.35; fuel and lime, \$213.75; total, \$1,414.10, or \$235 per month, about \$1 per month per tenement. The fuel used during this period was soft coal, but generally enough refuse lumber from the saw mill, or discarded concrete forms or similar waste is available to answer the purpose. This was largely so in 1913, and no coal has been burned since July, 1914.

Two one-horse wagons are used for collecting the refuse. They are low slung to make loading easier. One man drives each wagon, but the rigs travel together when collections from privies are made, as the loaded boxes are too heavy for one man to handle. A third man fires the incinerator and keeps the yard in proper shape, burning the garbage in heaps. These three men with the two horses comprise the operating force.

A number of concrete septic cesspools have been built, each taking the waste from one or two tenements; the water discharged is clear, with no odor. The simplest form has no partition. The inlet pipe extends to within 2 ft. of the bottom, and the outlet pipe turns down inside about 1 ft. below the level of discharge. This keeps the scum on the water undisturbed and allows the microbes to do their work. If there is any odor from the discharged water it usually indicates that too great a volume of water is going through, possibly due to a leak in the water system.

A welfare worker is employed, who is a trained nurse and is connected with the hospital service. She visits the houses and reports, on a card furnished her for this purpose, any unsanitary conditions or cases of illness. One of her most important duties is to give suggestions as to the care and feeding of infants and small children. This line of work is very helpful and does the greatest good.

When the results of this work get as far as the schools, the company is about in the same position as the old woman who lived in the shoe. We have to provide an additional school-room and teacher about every year. There are 542 children of school age, and nearly as many more younger. The present number of men working is 600.

Housing

The total number of houses occupied is 238. Of these, 90 take water from mains, seven from wells with individual electric pumps, and 141 from wells or cisterns. There are 21 houses with toilets and baths, and 217 with privies.

The older wooden houses are mostly of the double tenement type, with four or five rooms for each family. During the past eight years no wooden houses have been built. Instead, concrete blocks made from the tailings of the concentrating plants have been used. The interior floors and partitions are wood, and the roofs of slate. The concrete blocks are 8 by 10 by 20 in.; each occupies $1\frac{1}{10}$ cu. ft. of wall space. It costs 15c. to make a block, mixed 1 part cement and 6 parts tailings, this cost including sprinkling and piling for seasoning; 10c. additional covers the expense of hauling, handling, and laying in the wall, making the blocks cost 25c. each in the wall. No charge is made for the tailings, and the blocks are made at the foot of the tailings pile. The blocks are mixed



FIG. 4.—DOUBLE HOUSE OF CONCRETE BLOCKS AT MINEVILLE.

and tamped by hand, and made in a Century block machine. The 10-in. blocks have a 4-in. air space in the middle. On one of the first houses the experiment was tried of plastering the inside wall directly on to the concrete blocks, but this was a failure, as the water ran down the walls in streams; not due to moisture coming through the concrete-block wall, but to the condensation of moisture from the heated air inside coming into contact with the cold surface of the block wall. To remedy this trouble 1-in. strips were nailed to the blocks, to which wooden laths were nailed and the wall was replastered.

There are 88 concrete tenements of various types, single, double, and

four-family houses, including five larger boarding houses. The exterior and general arrangement of one of the double concrete houses are shown in Figs. 4 and 5. Most of those for American families are single houses with 50 to 60 ft. front to each lot, affording space for lawn and garden.

These concrete houses present a good appearance, are warm in winter, cool in summer, and are not damp. The cost of maintenance of the outside is practically nothing; inside it is the same as a wooden house.

The cost is about 10 per cent. more than wood. The secret of avoiding the sameness of appearance which spoils the effect of most concrete-block structures is in selecting the material to put in the face of the mold.

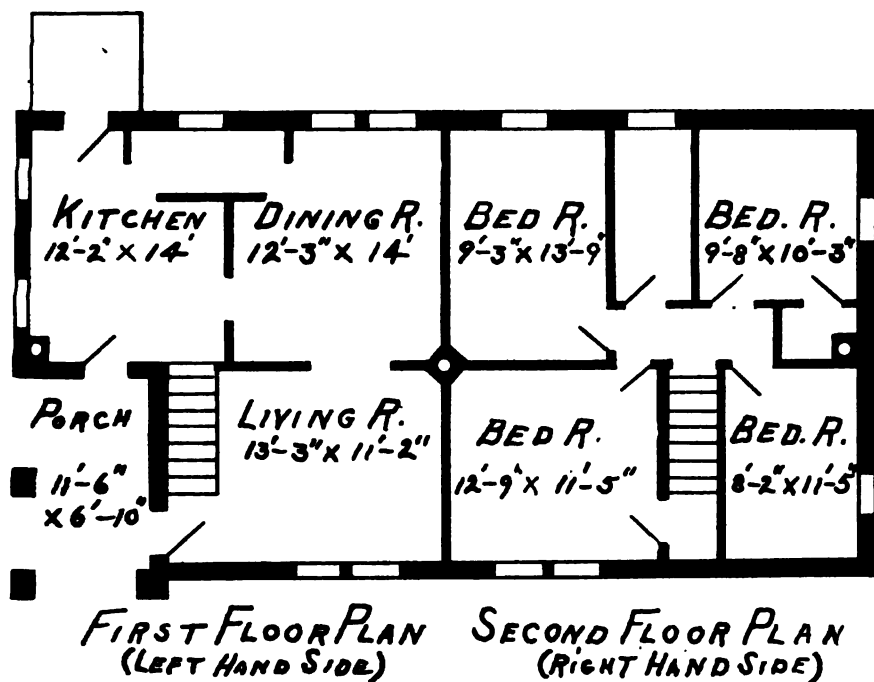
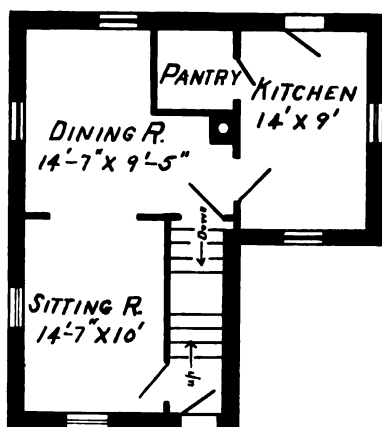


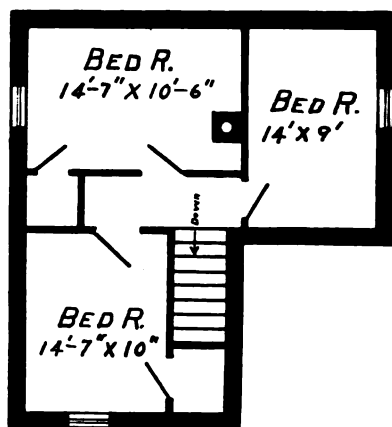
FIG. 5.—FLOOR PLANS OF HOUSE SHOWN IN FIG. 4.

If the face of one block is of moderately coarse material and the next one is all fine, when they are laid in the wall side by side the monotony is broken.

For the foreigners four-family tenements have proved quite satisfactory. These houses have a kitchen, a dining room and a family bedroom on the first floor, and two or three bedrooms on the second floor. Each family is a recruiting center, for when more men are wanted, they write their friends to come and get work and board with them. The larger boarding houses, accommodating 50 boarders, are not so popular, the men preferring to stay with the families who can take only five or six.

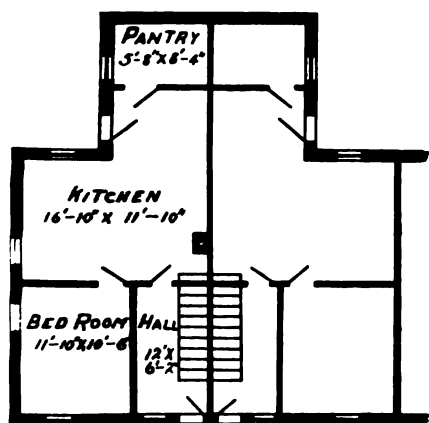


FIRST FLOOR PLAN

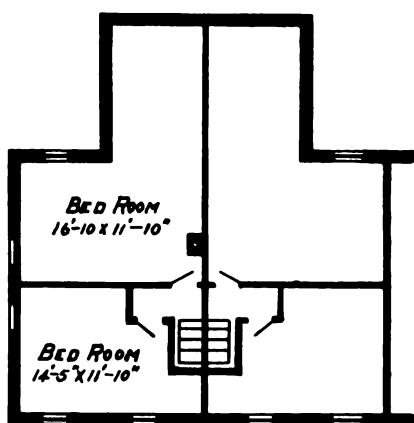


SECOND FLOOR PLAN

FIG. 6.—SINGLE HOUSE WITH SIX ROOMS; NO HEAT OR PLUMBING; SHINGLE ROOF. COST \$950, AND INDIVIDUAL FRAME BARN, \$100. RENT \$8 PER MONTH, INCLUDING BARN.



FIRST FLOOR PLAN



SECOND FLOOR PLAN

FIG. 7.—FOUR-FAMILY TENEMENT FOR FOREIGN LABORERS. SIZE 70 BY 26 FT.; CELLAR 70 BY 12 BY 7 FT.; SLATE ROOF. COST \$3,000, OR 6c. PER CUBIC FOOT. RENT \$5.50 PER MONTH, INCLUDING BARN.

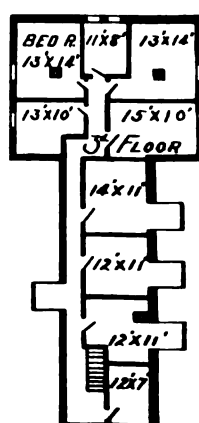
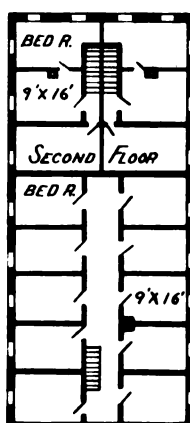
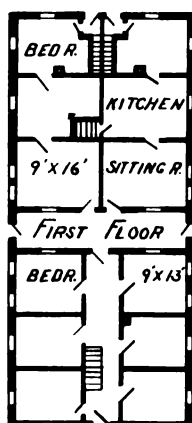
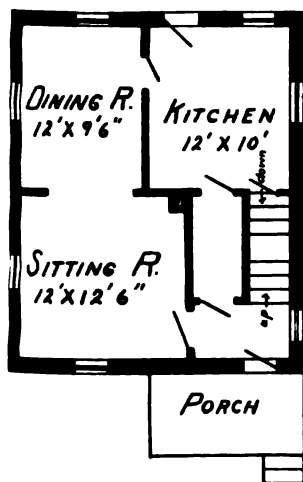
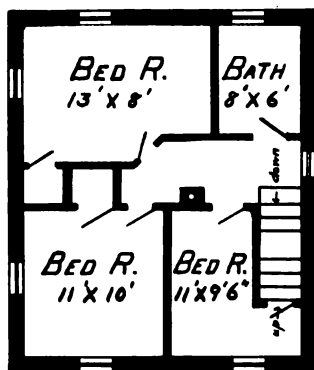


FIG. 8.—BOARDING HOUSE TO ACCOMMODATE 50 MEN, AND TWO FAMILIES TO KEEP THE HOUSE. COST \$4,000, WITHOUT HEAT OR PLUMBING. RENT \$25 PER MONTH FOR WHOLE HOUSE. IN HOUSE SHOWN IN PICTURE, STEAM HEAT AND PLUMBING, LAUNDRY AND WASH ROOM WERE INSTALLED AT COST OF \$1,000 ADDITIONAL.



FIRST FLOOR PLAN



SECOND FLOOR PLAN

FIG. 9.—ROW OF SIX SINGLE HOUSES, SIZE 26 BY 22 FT., TWO STORIES AND ATTIC, FULL-SIZED CEMENTED CELLAR. BATHROOM, BUT NO HEAT; WATER SUPPLY FROM WINDMILL AND WELL. COST \$1,350, OR 9c. PER CUBIC FOOT. RENT \$9 PER MONTH, INCLUDING BARN. EACH HAS A DIFFERENT COMBINATION OF ROCK-FACED AND PLAIN BLOCKS.

Each tenement has a small flower garden for each family, and a space of 30 to 40 ft. is left between the buildings. Barns are found to be essential and two double barns are built for each four-family house, with accommodations for a cow, chickens, and a pig. The privies are built in a corner of the barns. This general arrangement avoids a nondescript collection of shelters in each back yard. Prizes given for the best-kept lawn, flower bed, and window box have stimulated interest and pride in appearances, and have added greatly to the attractiveness of the village.

The following regulations apply to the keeping of live-stock: Cows and horses must be kept in barns or pastures, and out of yards and off the streets. Pigs must be kept in barns or pens at all times. Fowls must be kept on owners' premises. Yards, barns and all buildings must be kept clean at all times. Any live-stock or fowls found roaming in yards or streets will be put in the pound and released only on payment of a fine. A herder is employed by the company.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Underground Haulage by Storage-Battery Locomotives in the Bunker Hill & Sullivan Mine

BY J. W. GWINN, KELLOGG, IDAHO

(New York Meeting, February, 1915)

THE underground haulage system in the lead-silver mine of the Bunker Hill & Sullivan Co., situated at Kellogg, Idaho, is the most extensive in the Cœur d'Alène district, comprising about 35,000 ft. of tunnels, drifts, and cross-cuts where locomotive haulage is in use. The main working adit, known as the Kellogg tunnel, or No. 9 level, is approximately 10,000 ft. in length from the portal to the main working shaft and extends beyond that about 4,500 ft. in each of two directions. On this level all the ore from the Bunker Hill & Sullivan, the Sierra Nevada Consolidated, and the Caledonia Mining Co.—an average of 44,600 tons per month—is hauled to the mills, located a short distance from the portal. Electric haulage by the trolley-type locomotive is used on this level throughout.

Below this level, at intervals of 200 ft. vertically, are five other working levels (Nos. 10, 11, 12, 13, and 14); and the total haulage distance on these, where electric locomotives are used, is approximately 16,000 ft. No. 13 level has but recently been opened up and is not working to capacity; while the main cross-cut on No. 14 has just been started, and has no need of power haulage. Of the four levels equipped with electric haulage, Nos. 11, 12, and 13 have storage-battery locomotives, and No. 10 still retains the trolley type by reason of the small and decreasing tonnage produced.

Electrification

Electric power is supplied to the mine by two systems. The first, the 500-volt direct-current trolley, was installed in 1897 when the Kellogg tunnel was started, and was extended as the tunnel advanced, being used for the first trainload of ore on Nov. 17, 1902, since which time it has been in continuous service. The current for the line is transmitted on a No. 00 copper wire as far as the shaft, and on No. 0 wire throughout the remaining length of the Kellogg tunnel; and until the installation of the storage-battery locomotives it was used in practically all the cross-cuts and drifts on each level below No. 9.

The second circuit, 2,300 volts, alternating current, was installed in 1907, to furnish power for the pumps, as well as lights for the shaft and stations, and for preheating compressed air for the hoist. This current is transmitted on a No. 2 B. & S. gauge, triple-conductor, varnished cable incased in a lead sheath. The installation of an electric hoist in November, 1911, increased the load on the cable so much that an additional cable, No. 000, leaded, varnished cambric, covered with jute, was added, as an additional protection in case of cable break-downs. The two cables are kept continuously in service, so that any of the motors requiring 2,300-volt three-phase service can be supplied from either of the two. Also, any break-down in insulation, or other injury, can be immediately detected and repaired before any possible call can be made on either of



FIG. 1.—2¼-TON JEFFREY LOCOMOTIVE ON NO. 11 LEVEL, SHOWING THE ARRANGEMENT OF THE INTERCHANGEABLE AUXILIARY BATTERY OF CELLS.

them for the full load of the pumps and hoists combined. This practically insures continuous service so far as the transmission of current underground is concerned. In addition to being used for the purposes stated above, these high-voltage lines also furnish current for the motor-generator set used in charging the storage batteries.

Storage-Battery Installation

The first storage-battery locomotive to be installed, replacing a 2½-ton Jeffrey trolley-type, was a Jeffrey, put into operation Mar. 3, 1913, on No. 11 level, of the following specifications:

Weight.....	5,000 lb.
Maximum volts required at battery terminal for charging.....	125
Average volts at battery terminal on discharge.....	75
Number of motors.....	2
Wheels inside or outside.....	inside
Approximate full-load speed, per hour.....	4 miles
Track gauge.....	24 in.
Overall width.....	40 in.
Length of frame.....	83 in.
Height of frame above rail.....	24½ in.
Height over battery.....	46 in.
Wheel base.....	27 in.
Diameter of wheels.....	16 in.

The battery equipment consists of 63 cells of Edison type "A 4," having a capacity of approximately 112 ton-miles on a single charge. Ampere-hour capacity, 150.



FIG. 2.—AUXILIARY BATTERY OF CELLS FOR 2½-TON JEFFREY LOCOMOTIVE BEING CHARGED AT THE STATION ON NO. 11 LEVEL.

The locomotive was equipped with two batteries, so that one could be left at the charging station to be charged, while the other was on duty. This was made necessary by the insufficient capacity of one battery to run for the required time without loss of time for recharging (Figs. 1 and 2).

The second installation, replacing a 4½-ton General Electric trolley-type, was made on No. 12 level, June 8, 1913, when a Westinghouse storage-battery type (Fig. 3) of the following specifications was put into service:

Weight.....	7,600 lb.
Full-load speed, per hour.....	3.6 miles
Running drawbar pull on clean dry rails.....	1,920 lb.
Starting drawbar pull with sand.....	2,400 lb.
Gauge.....	24 in.
Diameter of wheels.....	24 in.
Wheel base.....	3½ ft.
Height over all, exclusive of trolley.....	3 ft. 10 in.
Length over all, exclusive of bumper blocks.....	9 ft. 6 in.
Motors.....	2
Battery, Edison type "A 8," consisting of 70 cells, with a rated discharge capacity of 300 amperes.	

The third installation was made May 11, 1914, on No. 13 level, with a General Electric storage-battery locomotive of the following specifications:

Weight, including battery, approximately.....	8,000 lb.
Rated drawbar pull on level track.....	2,000 lb.
Speed at rated D.B.P., per hour.....	3 miles
Voltage.....	72
Gauge.....	24 in.
Overall length, not including couplers.....	10 ft. 3 in.
Overall width.....	42 in.
Height over platform.....	29½ in.
Height over battery compartment.....	46 in.
Wheel base.....	38 in.
Diameter of wheels.....	20 in.
Motors.....	2
Battery consists of 70 Edison type "A 8" cells having a rated discharge capacity of 300 amperes. The line voltage necessary for full charge must be at least 130 volts D. C.	

Each of the last two locomotives has only a single battery of cells, of sufficient capacity to run on full load for the required length of time.

Charging

For charging purposes, a motor-generator set, 125-volt, three-phase, 50-h.p. motor, located in the hoist room on No. 9 level, takes current from the 2,300-volt A. C. line and delivers 125 volts, D. C., to the charging stations on each level. It was designed for the ultimate capacity of storage-battery locomotives on all levels from 10 to 14 inclusive.

A wattmeter is cut into the line at the motor-generator set, recording the amount of current delivered to it from the 2,300-volt A.C. line. Of course, there are losses, which occur in the motor-generator set, in the transmission of the current from it to the batteries, and from the batteries to the motors; but the total kilowatt-hours consumed is taken from the wattmeter, so the costs figured on that basis include these losses.

The motor-generator set is supplied with overload circuit breakers and reverse-current relay, which protect it from overload as well as from

the possible reversal of current due to low voltage or stopping of the motor generator while charging.

Rheostats are provided at each charging station so that both voltage and amperes can be delivered to the battery as required, as locomotives are sometimes called upon for extra heavy duty during one shift and the charging current during the interval between shifts is often made at twice the normal rate.

After some practice it was found that the two locomotives supplied with a single set of batteries each, would operate successfully for two shifts on the following charging schedule and not give any signs of weak batteries.

Schedule for Charging—Starting at 8:00 a.m.

Operate without charging.....	4 hr.
Charge.....	30 min.
Operate without charging.....	4 hr.
Charge.....	2 hr.
Operate without charging.....	8 hr.
Charge.....	5 hr.
Idle without charging.....	30 min.
Total.....	24 hr.

Performance

It is not the intention here to make any comparison of the merits of the different types of storage-battery locomotives, but simply to give facts, as we have been able to get them, on the performance of these locomotives under the working conditions as they are here. Two of the locomotives, the Jeffrey on No. 11 level and the Westinghouse on No. 12 level, haul trains made up of seven and nine cars respectively, of 34 cu. ft. capacity each, or approximately 5,000 lb. of ore per car, on a track whose average grade is one-half of 1 per cent. in favor of the load, and are working full time. The General Electric on No. 13 level has only a light tonnage to handle and consequently does not work full time. This level has only been opened up recently and has not reached a normal output. But in collecting data for this article, it was impossible to separate the current consumption for the three locomotives, so that these costs are figured collectively for the three, with the average tonnages and distances hauled for each level figured separately and then averaged for the three. As will be seen, the costs for the General Electric locomotive, in kilowatt-hours consumed per ton of ore hauled, are much higher than for the other two. This is due to the fact that part of the ore on that level is trammed by the shovelers, making it impossible to get exact tonnages hauled by the locomotive. Also, most of the work of this locomotive has been in hauling waste, of which no account has been kept, so that the actual tonnage hauled is much greater than that stated.

The average kilowatt-hours delivered through the motor-generator set to the charging circuit for all three locomotives, taken from the watt-meter readings over a period of seven months, is 6,845 kw-hr. per month, and the total average ore tonnage hauled by the three locomotives is 27,660 tons per month, which gives a kilowatt-hour consumption of 0.2474 kw-hr. per ton of ore hauled.

The average distance that all three locomotives pull a loaded train is 730 ft., which gives 0.0339 kw-hr. consumed per ton of ore hauled per 100 ft., or for the round trip, $\frac{0.0339}{2} = 0.01695$ kw-hr. per ton per each 100 ft. the train is moved.



FIG. 3.—4-TON WESTINGHOUSE STORAGE-BATTERY LOCOMOTIVE ON No. 12 LEVEL, SHOWING THE TYPE OF CAR USED FOR HAULAGE PURPOSES.

These figures are compiled on the average tonnages taken over a period of five months, but do not represent the full duty of the locomotives, since each level has a considerable tonnage of waste rock to be transferred every day, and consequently the kilowatt-hour consumption per ton of all material moved is appreciably less.

Efficiency

As has been said, losses occur in the transmission of current from the A. C. lines to the motor-generator set, and from the generator set to the batteries, and again in the transmission from the batteries to the motors. The normal voltage of the cells is approximately 110 volts, but, owing to a

peculiarity, characteristic of the Edison cell, drops to about 75 volts when the load is applied to the motor.

The efficiency and drawbar pull of mining locomotives are usually calculated by the manufacturer on the basis of clean dry rails and true roadbed; but this condition does not prevail in most mines, and especially in the Bunker Hill & Sullivan, where the tracks are often on the sill floors of stopes. For this reason, the roadbed is rough and irregular and has many short curves, and, moreover, overhanging chutes are continuously dropping small rock and water directly on the rails, which has to be crushed by the car wheels on each return trip and thereby greatly increases the normal drawbar pull required.

Maintenance and Repair

Since the installation of the storage-battery locomotives, the repairs on the motors and batteries have been practically nothing. The low voltage, as compared to the 500-volt D. C. used on the trolley-type locomotives, practically eliminates brush and commutator troubles, which always have been a source of heavy expense. About the only charge against the batteries is the time of one man for a few minutes each morning, giving them the daily inspection and refilling the cells with distilled water to replace that evaporated during the previous day—the amount of distilled water required for three batteries being about 20 gal. per week. In addition, there is a monthly charge, not exceeding \$10 per battery, for cleaning and general overhauling. The principal source of repair expense on the locomotives to date has been new wheels.

The figures given below are the total average monthly cost of repair and upkeep from the date of installation to Nov. 1, 1914, and also include the cost of installation, which was quite large, because the battery boxes had to be altered and partly rebuilt, to adapt them to our charging system, and to protect them from the water issuing from the chutes under which they pass.

Storage-Battery Locomotive

	Average Monthly Repair Cost	Average Monthly Tonnage Hauled	Repair Cost Per Ton
2½-ton Jeffrey, No. 11 level	\$48.513	13,501	\$0.00359
4-ton Westinghouse, No. 12 level . . .	55.456	12,755	0.00434
4-ton Gen. Electric, No. 13 level . . .	28.612	2,406	0.01189

The last figure is high because the costs are figured only on the tonnages of ore hauled, and most of the material hauled by this locomotive has been waste.

A comparison of these costs with those of the trolley locomotives which they replaced may be interesting. They cover a period of two

months in the first case and four months in the second, so that the figures must not be taken to represent an average cost over a long period. Separate repair costs for all locomotives were not kept until January, 1913, which accounts for the short period taken for the above locomotives, when it is remembered that they were replaced by storage-battery locomotives in March and June, respectively, of the same year.

Trolley Locomotive

	Average Monthly Repair Cost	Average Monthly Tonnage Hauled	Repair Cost per Ton
2½-ton Jeffrey, No. 11 level.....	\$39.88	9,154	\$0.00435
4½-ton Gen. Electric, No. 12 level...	93.912	14,645	0.00641

These figures do not include the initial cost and upkeep of the trolley wires and track bonding, which kept two men busy practically all the time, and which consequently was a heavy expense. No separate costs were kept for these levels, however, so they must be omitted in this connection.

It has been estimated by the company's electrical engineers that, with a few minor improvements in the charging system, and a better understanding of, and more careful attention given to, the operation of these locomotives by the motormen, the costs of repair and operation will be 75 per cent. less than for the trolley locomotives doing the same work.

Advantage of Storage-Battery Haulage

Different conditions in different mines make the haulage problem one to be worked out to suit individual needs; yet some of the advantages of storage-battery haulage will apply to all; and others, as we have found them here, may be of interest to those who are contemplating the installation of a haulage system.

First of all, storage-battery haulage does away with the dangerous trolley wire. Many of our drifts are on the sill floors of the stopes, where the timbers are depressed and often broken by the weight of the filling above, so that there is always danger of the employees coming in contact with the low-hanging, high-voltage trolley wire. This is not only dangerous to the train loaders and the repair men in the drift, but equally so to the miner walking through the drift, or climbing up a manway with his tools and drill steel. During the 17 years that the mine has been equipped with electric haulage, there have been three fatal accidents caused by contact with the trolley wires.

Another important feature is the easy access to the face of a drift with the storage-battery locomotive. There is no trolley wire to blast

against and break, with the attendant delay of finding the electrician to make the necessary repairs; neither is it necessary to wait for him to extend the trolley wires and track bonding as the face advances.

As mentioned in a previous paragraph, the cost of installation and upkeep of trolley wires and track bonding is a big item; and additional to that is the cost of replacing broken trolley poles and trolley heads, and the delay occasioned thereby.

Trolley-locomotive haulage necessitates provision for the return of the grounded current. Here that is done by a wire connection between the rail of the drift and a rail in the shaft. This connection has become broken several times, and consequently the current has been conducted by means of the wet board flooring to the iron pipes of the pumping system. In two instances the resulting electrolysis made a hole through the intake pipe of the pump, preventing it from drawing water.

These are only a few of the more important advantages of storage-battery haulage in underground practice, and there are many minor ones that could be added; but they may all be summed up in three words—safety, economy, and efficiency.

The management and the electrical staff are thoroughly satisfied with the results obtained from the installation, and are firmly convinced that with the care of the batteries better systematized and in the hands of competent, conscientious men, the ultimate results will exceed their original expectations.

Acknowledgments are due to W. C. Clark, Electrical Engineer, and M. J. Bottinelli, Assistant Electrical Engineer, of the Bunker Hill & Sullivan Mining & Concentrating Co., for their assistance in collecting data for this paper.



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The Hydro-Electric Development of the Peninsular Power Co.

BY CHARLES V. SEASTONE,* MADISON, WIS.

(New York Meeting, February, 1915)

Location

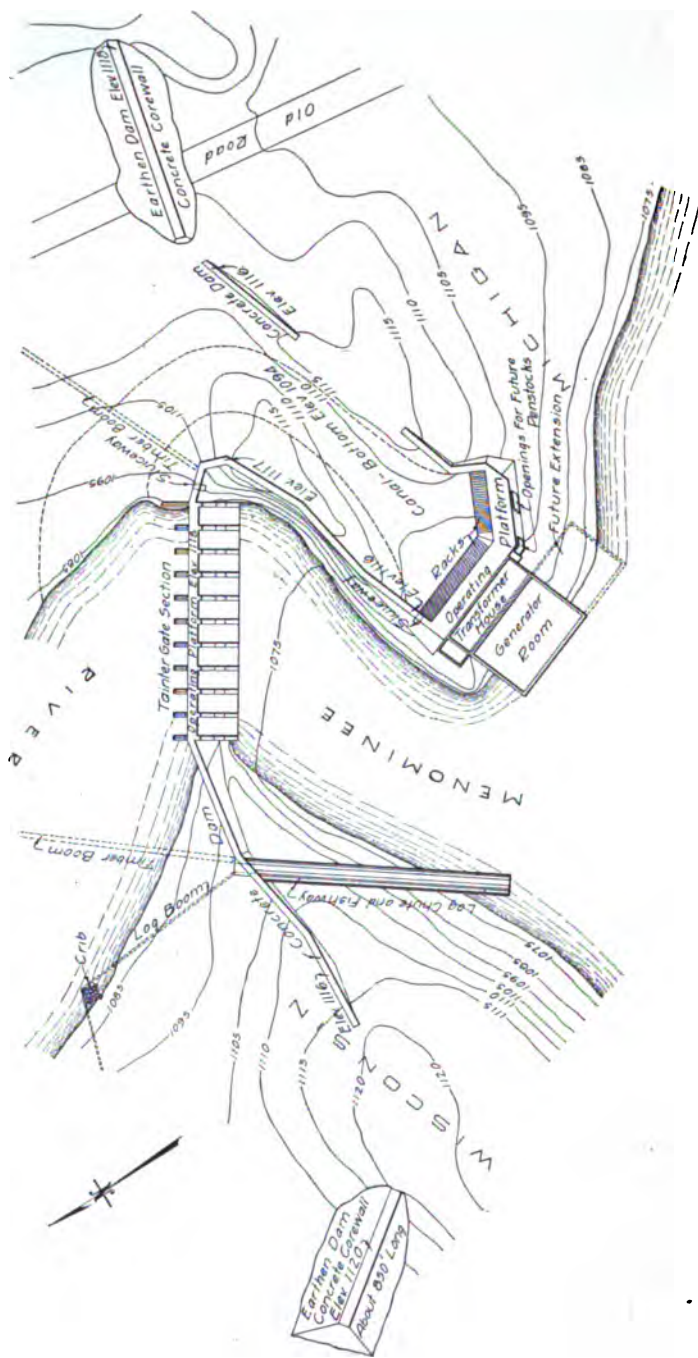
THE hydro-electric plant of the Peninsular Power Co. is located at what is commonly known as Lower Twin Falls on the Menominee River. This location is about $3\frac{1}{2}$ miles north of the city of Iron Mountain, Mich. The principal points of delivery of electric energy from this plant are in the mining regions adjacent to Florence, Wis., and Iron River, Mich., and in the cities of Iron River and Iron Mountain. The hydro-electric plant at Twin Falls and the substation of the Iron Mountain Electric Light & Power Co. are connected by a 6,600-volt transmission line carried on steel towers and steel poles, the latter being used within the city limits. The plant at Twin Falls and the substation at Iron River, about 36 miles distant, are connected by a 66,000-volt, duplicate, three-phase transmission line, supported on steel towers. At Iron River, are located a 1,500-kw. auxiliary steam plant, and the substation, from which the current is distributed to the mines in this vicinity. A description of the auxiliary plant, main and customers' substations, and of the main and secondary transmission lines, follows later in this paper.

General Layout

The relative location of the dam, forebay, and power house will be understood by referring to Fig. 1. The dam is constructed on a ledge of solid rock which forms Lower Twin Falls. This rock ledge offers an excellent location in that the height of the dam, and therefore the amount of concrete, are thereby reduced, and also in that practically no pumping was required in unwatering the dam site. The maximum head provided by the dam is 44 ft., with an average working head of from 40 to 42 ft.

The other structures, including the substructure of the power house,

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the forebay or canal section through which the water is carried to the penstocks, and the head-gate section containing the trash racks and penstock gates, are founded on solid rock. The general plan of development shown in Fig. 1 was adopted, (1) because it afforded a location for the power house which is practically free from any serious effects

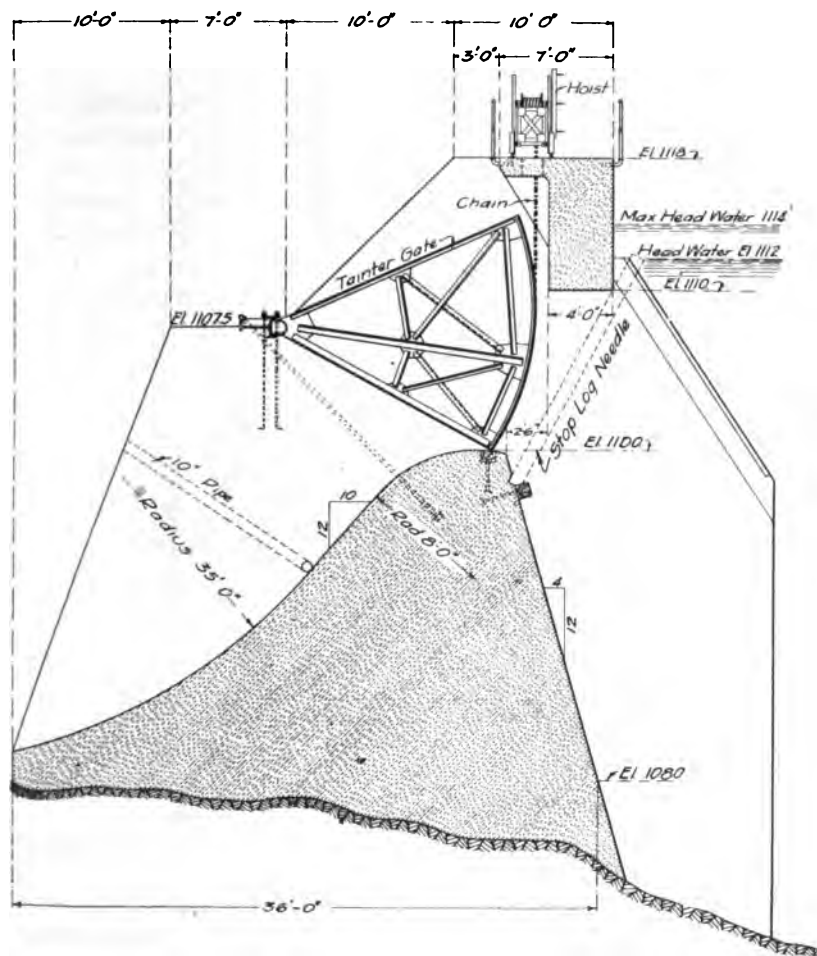


FIG. 2.—CROSS-SECTION OF DAM.

of floating materials, and (2) because of the ease of construction of the plant. The canal section is not lined except for a short distance where it joins the head-gate section, and is practically free from seepage.

Dam

The portion of the dam crossing the river proper consists of a Tainter-gate section for passing the flood flow of the river, consisting of 10

Tainter gates, each 14 ft. high and 14 ft. long. Fig. 2 is a cross-section through the Tainter-gate section and illustrates the general arrangement, design, and method of anchorage of the gates. This section is constructed of solid concrete throughout, and is of the gravity type, provided with a solid ogee section for the spillway. The Tainter- or flood-gate section of the dam, which is about 170 ft. long, is flanked on the Wisconsin side by a gravity-section concrete dam founded on solid rock. In addition to this, on the Wisconsin side there is an earthen section about 14 ft. high, and about 850 ft. long. This earthen section is 8 ft. wide on top with side slopes of 3 to 1 on the water side and 2 to 1 on the inside. This earthen section is provided with a solid concrete core wall, well anchored to the solid rock which underlies the surface. All of the rock in this vicinity is of a granitic formation, commonly known as greenstone. In addition to the flood gates, the dam is provided with a fishway of a type approved by the Wisconsin and the U. S. Fish Commissions. Next to the fishway is a chute to provide for the passage of logs or the floating of débris. Fig. 3 shows the general

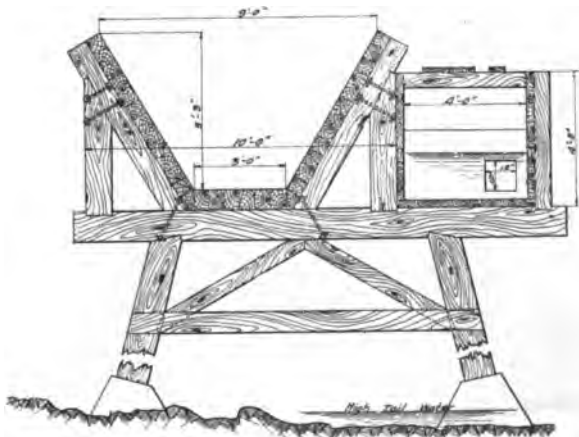


FIG. 3.—SECTION OF LOG SLUICE AND FISHWAY.

arrangement of the log chute and fishway and the general plan of construction. Suitable booms, anchored to heavy rock-filled cribs, extending to the bed of the river, are constructed above the dam for guiding the logs to the chute.

On the Michigan side, in the base of the Tainter-gate section, is a 4 by 4 ft. sluice gate, operated by a stand on the operating platform. This gate is utilized in the winter season when it is sometimes desirable to discharge only a small amount of water, and when, due to ice conditions, the operation of the Tainter gates is more or less inconvenient and difficult. The Tainter gates are operated by a hand winch which moves on a track on the operating platform.

Power House

The superstructure of the power house is constructed for the present installation of three 1,000-kw. generating units and the substructure has been completed for the two additional 1,000-kw. units which are to be installed in the near future. The head-gate section and canal or forebay leading thereto are also completed for the final installation of 5,000 kw. Figs. 4 and 5 show the general arrangement of the power house and head-gate section and the general arrangement of the switchboard, hydraulic, and electrical machinery.

The head gates are constructed of timber having steel bearing plates, and each is operated by a substantial individual hoist. Filler gates are provided in the penstock gates, although the hoists are designed to move

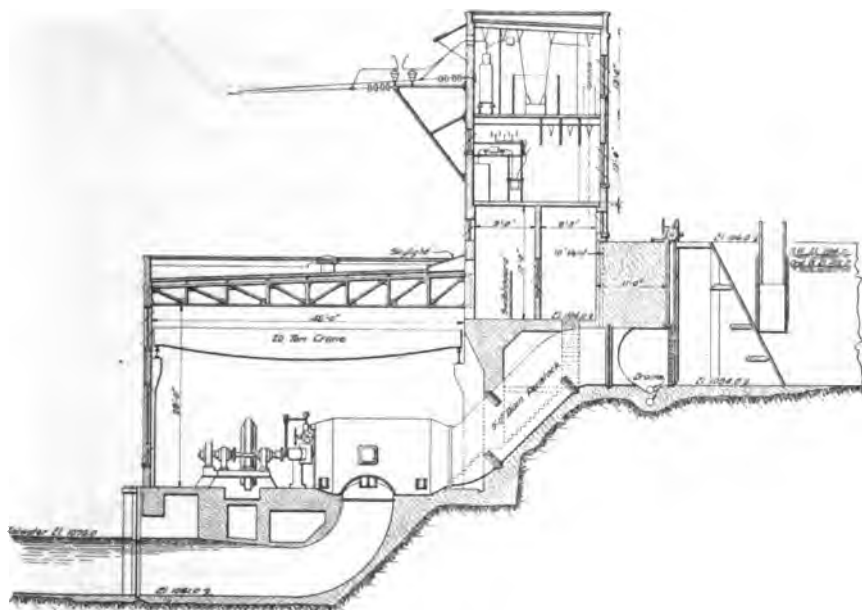


FIG. 4.—SECTIONAL ELEVATION, TWIN FALLS PLANT.

the latter without relieving the pressure. At the end of the trash racks is a sluice gate with a movable crest which provides means to sluice away ice or other débris that may collect in front of the racks. The latter are constructed from a special section, providing for a large entrance area, thereby reducing entrance losses and losses due to velocity head. The racks are so constructed as to admit of being easily raked and kept free from débris.

In the design of the hydraulic features of the plant, care was taken to provide liberal areas for waterways, and so arrange the plant as to minimize the labor necessary for its successful and economical operation.

The superstructure of the power house consists of a three-story brick building containing the switchboard gallery, individual transformer compartments provided with steel doors, high-tension switch compartments, etc., the general arrangement of which can be best seen by referring to Figs. 4 and 5.

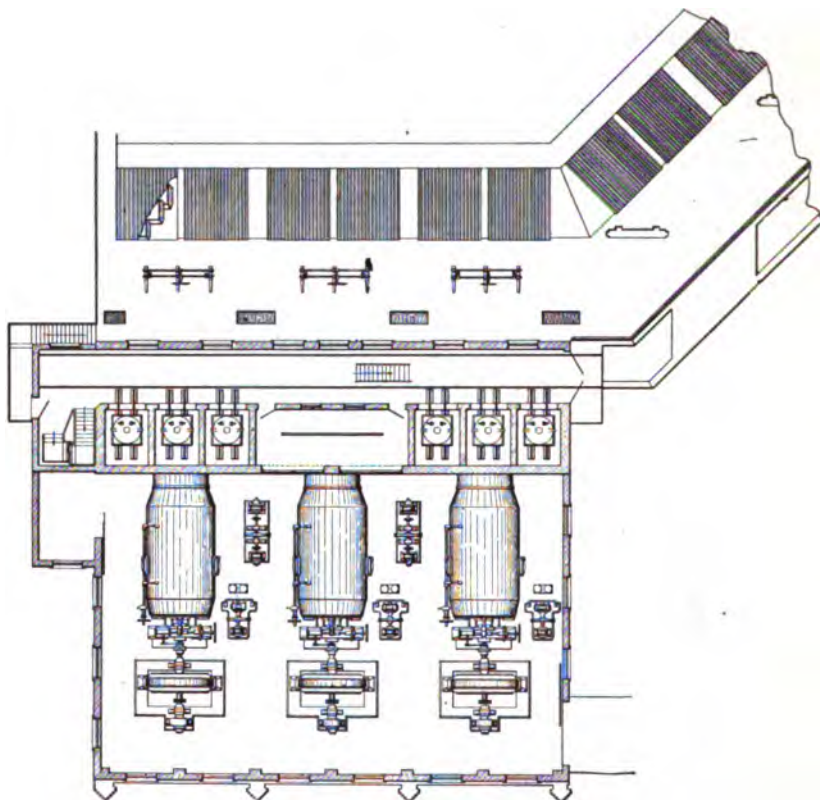


FIG. 5.—SECTIONAL PLAN, TWIN FALLS PLANT.

Stream Flow

The Menominee River forms part of the boundary between Wisconsin and the upper peninsula of Michigan. It is formed by the junction of the Brule and Michigamme Rivers, and flows in a southeasterly direction into Lake Michigan through Green Bay. A gauging station was established at the Homestead highway bridge about $2\frac{1}{2}$ miles south of Iron Mountain by the U. S. Geological Survey, in September, 1902, and records are available from that time to date. Discharge measurements are made from the single span to which the gauge is attached. The rating of this section, that is, the relation between gauge heights and discharge, has been carefully checked, and the relationship as established

by the government has been found correct. The section of the river at this rating station is favorable for good results both as to straightness of channel and permanency of bottom.

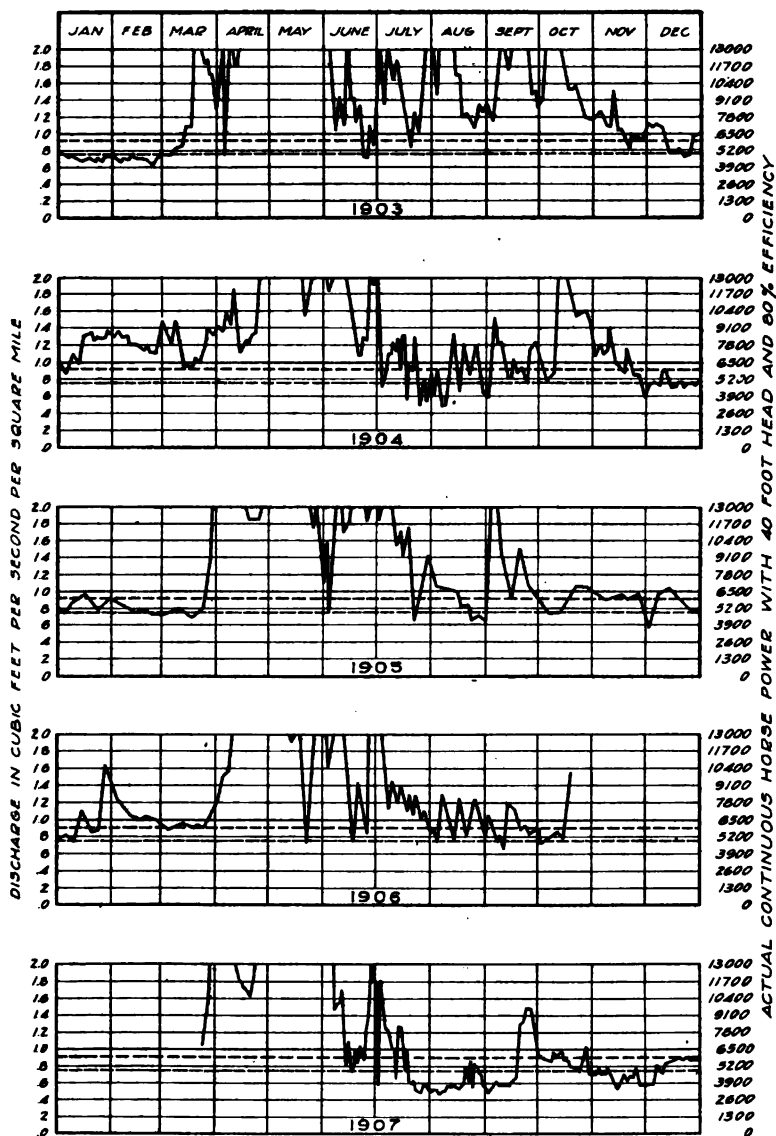


FIG. 6.—HYDROGRAPHS OF THE FLOW OF THE MENOMINEE RIVER, 1903 TO 1907.

Figs. 6 and 7 show hydrographs of the daily flow of the Menominee River for the years 1903 to 1912, inclusive, and are platted from the government gaugings referred to. The left-hand scale of these hydro-

graphs shows the discharge in cubic feet per second per square mile, which, if multiplied by the drainage area (1,790 square miles) above Twin Falls—the site of the hydraulic plant—and compared with the

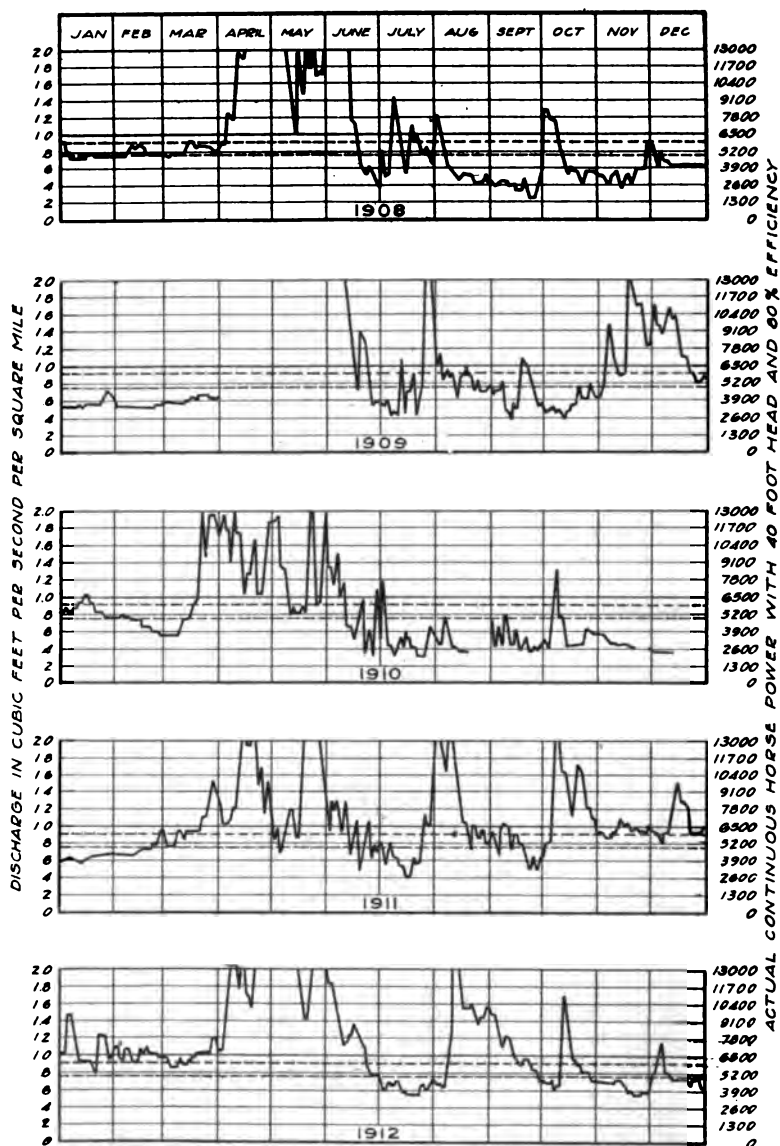


FIG. 7.—HYDROGRAPHS OF THE FLOW OF THE MENOMINEE RIVER, 1908 TO 1912.

irregular line of flow, will give the total discharge for any day of any year above mentioned. The right-hand scale shows the continuous horse-power at the turbine shaft with 40 ft. head and 80 per cent. efficiency.

This scale when compared with the irregular line of flow will give the horsepower at the turbine shaft for any day of any year for which records of flow are available. Inasmuch as it is possible to obtain a gross head of 44 ft. during periods of low flow, it is believed that the head used as a basis for calculating the amount of power will more than compensate for any losses that will occur. An examination of these hydrographs shows that, by the use of the pondage that is available above the dam, 2,500 continuous horsepower could have been delivered by the turbines during each day except for a short period during the years 1908, 1909, and 1910. This corresponds to about 1,500 continuous kilowatts delivered to the customer, allowing for proper losses in generators, transformers, and line.

With the 1,500-kw. steam-turbine plant at Iron River, the hydraulic output at Twin Falls can at low water be increased to about 5,000 h.p. The lower heavy line shown on each hydrograph represents the combined hydraulic and steam output with the present steam and hydraulic plant, and represents about 2,900 continuous kilowatts delivered to the customer.

There is also shown on the hydrographs an upper heavy horizontal line which corresponds to about 3,400 continuous kilowatts delivered to the customer. To deliver this continuous output and provide for the peak load would necessitate the full hydraulic equipment at Twin Falls of five 1,000-kw. units, one of which would constitute a reserve unit, together with a 2,000-kw. steam plant at Iron River, or 500 kw. in addition to that now installed.

Hydraulic Machinery

For the present installation, three units are provided. Each unit consists of a pair of 40-in. Samson horizontal-shaft, center-discharge turbines. Each unit is mounted in steel case penstock, and direct connected to a 1,250-kva. generator. Each turbine unit operating under an effective head of 42 ft. will develop 1,700 h.p. at full gate, running at a speed of 257 rev. per minute. With each turbine unit is installed a system of positive lubrication, consisting of a large grease compressor which is fitted with suitable hand wheel and gears for operating it. Holyoke tests of a similar 40-in. Samson runner showed almost 89 per cent. efficiency at about three-quarter gate, and the manufacturer's guarantee for the wheel installed was 86.5 per cent. at about three-quarter discharge—i.e., when developing about 1,400 h.p. Under the conditions of installation it is believed that efficiencies somewhat greater than the manufacturer's guarantee will be realized.

For controlling the speed of these turbines three Lombard oil-pressure water-wheel governors are installed. These governors are of horizontal

design and are mounted on the bridge tree astride of the main turbine shaft close up to the flume head, thus occupying no floor space. With each of the governors is installed a motor-driven pump for 200-volt, three-phase, 60-cycle current. Individual pressure and vacuum tanks, also Lombard speed controllers for switchboard control, are likewise provided.

Generators

The generators are of Westinghouse make, and have a rating of 1,250 kva., 6,600 volts, 60 cycles, three phase, 257 rev. per minute, each equipped with a 120-volt direct-connected shunt-wound exciter of 27.5 kw. capacity, designed for operation with Tirrill regulator. The rotor of the generator is of the flywheel type, so designed as to assist materially in the speed regulation of the plant. All rotating parts are designed for 60 per cent. overspeed.

Transformers

The ultimate transformer installation will comprise six single-phase, 835-kva., oil-insulated, water-cooled transformers, arranged in two banks, delta-connected on the low-tension side, receiving 6,600 volts, and Y-connected on the high-tension side, delivering 66,000 volts. The present equipment consists of one bank of transformers with one extra single-phase unit acting as a spare.

Each transformer is placed on rails in a separate brick compartment closed by a steel rolling-curtain door. A transfer truck is provided to facilitate the moving of the transformers. In the west end of the transformer building an opening is left through all the floors large enough to allow the raising of the core out of the transformer with the aid of a triplex chain hoist attached to an I-beam on the upper floor. This opening was also used in hoisting material to the different floors during the construction of the plant. The cooling water for the transformers is supplied by two motor-driven centrifugal pumps, one of them being held for reserve.

A complete oil-piping system is installed, connecting each transformer with a Westinghouse oil-treating outfit of the filter-press type, so arranged as to permit cleaning the oil of any transformer without taking the transformer out of service.

Switchboard

The electrical energy generated is controlled by a complete switchboard. Fig. 8 is the wiring diagram for the generating station at Twin Falls. All oil switches are separately mounted and are controlled electrically from the main control board, located on the second floor between

the two transformer groups and overlooking the generator-room floor. All high-tension and low-tension measuring transformers are placed in fire-proof brick compartments, closed by ebony asbestos doors.

The individual busbar conductors of the high-tension, also of the low-tension system, are separated from each other by barriers of ebony asbestos wood. The 66,000-volt oil switches are of the horizontal break type.

Special care has been exercised in providing for an uninterrupted service, and the equipment installed for this purpose is unusually complete and may be classified as follows:

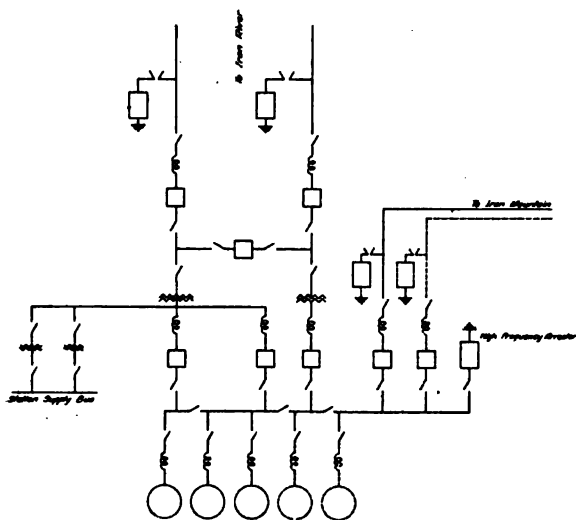


FIG. 8.—WIRING DIAGRAM, GENERATING STATION, TWIN FALLS.

- a. Each three-phase line is equipped with a modern type of electrolytic lightning arrester.
- b. In connection with the lightning arresters, each individual overhead conductor leaving the power house has been equipped with a choke coil, wound with Swedish iron to increase its resistance greatly under lightning oscillations.
- c. All the high-tension station wiring has been done with solid Swedish iron, for the same reason.
- d. All high-voltage current transformers have been protected by special electrolytic cells.
- e. A special new type of arrester has been imported from Europe, in addition to the American types already installed, for the purpose of eliminating troubles due to high-frequency disturbances by lightning, switching, arcing grounds, or otherwise.
- f. All circuits connecting with the busbars are provided with in-

dividual choke coils of Swedish iron to protect the machinery and station apparatus against high-frequency and short-circuit effects.

g. Two special motor-generator sets have been provided, each coupled to a heavy flywheel (laminated) of boiler-plate steel, to furnish emergency lighting and power to operate temporarily the oil switches in the very remote case of an accidental shutdown of the whole plant.

h. A special metallic grounding rheostat was installed between the transformer high-tension neutral and ground, together with a special current transformer, alarm bell, signal lamp, ammeter, and special relays. This equipment serves, in case of accidental ground on the transmission line, to limit the ground current to a moderate amount and keep the oil switches from opening automatically, thereby maintaining uninterrupted service notwithstanding trouble on the line. The rheostat is of such capacity as to give the station operator ample time to locate the line in trouble and either isolate same, or, more frequently in case of an arcing ground, to interrupt service over this one line for one second (maintaining service over the other line), and thereby remove the trouble.

Transmission Line

The transmission line leaves the power house through a large porcelain outlet bushing of the corrugated type, protected from the rain by a large concrete hood extending 5 ft. from the building wall. The line is built in duplicate and is supported on steel towers set in concrete bases. Two types of towers are used, namely, a flexible or two-post structure, the main members of which are channels, and a four-post tower constructed from angle sections, which serve as anchorages for the line. On all angles above 12° a heavier four-post structure is used. The percentages of the different types of towers are as follows: Standard two-post, 73 per cent.; standard four-post, 17 per cent.; angle towers, 9 per cent.; special towers, 1 per cent. Fig. 9 shows a standard two-post tower; Fig. 10, a standard four-post; Fig. 11, an angle tower; and Fig. 12 shows the special tower construction for the lines entering the substation at Iron River, Mich.

The concrete anchorages for the standard two- and four-post towers were constructed by molding concrete pedestals around the base or anchorage angles. The forms for these pedestals were constructed of galvanized iron and the work was done at convenient points along the line of construction, and the bases distributed to the site and set. The pedestals for the standard two- and four-post towers are 6 by 10 in. at the top, 9 by $11\frac{1}{2}$ in. at a point 1 ft. from the bottom, and 18 by 24 in. at the bottom. They were cast 4 ft. 8 in. long, and the anchorage iron projects about 1 ft. above the top. Where a line parallels a railroad, and transportation is therefore facilitated, the method above outlined

for forming and setting the concrete anchorages has been found to be more economical than the usual method of depositing concrete in the earth around the anchorage iron, and also gives excellent results. For the heavy angle towers the last-named method was employed.

The conductors consist of six aluminum wires, of a total cross-section equivalent to No. 0 B & S. gauge, around a high-strength galvanized-steel center, having an elastic limit of 130,000 lb. per square inch. The power cables are strung with a sag considerably greater than that recommended

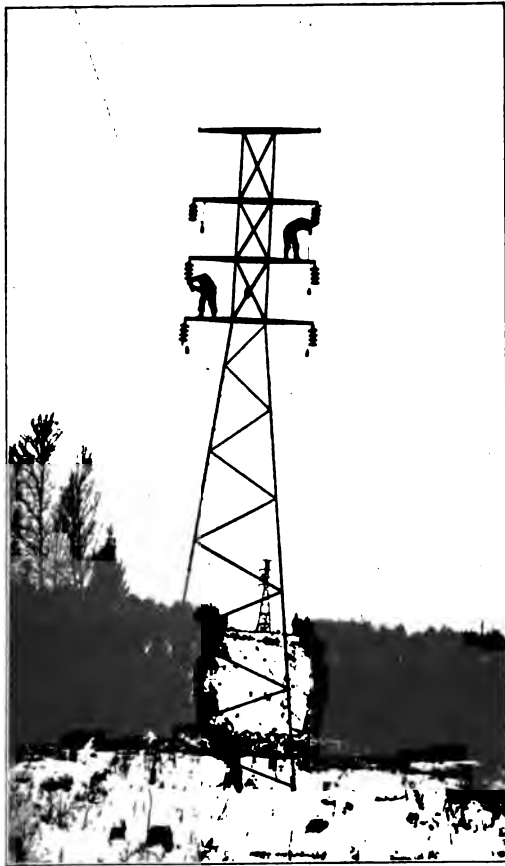


FIG. 9.—Two-Post Tower.

by the manufacturers and guaranteed by them as being entirely safe under the worst possible conditions of wind and sleet loads at low temperature. This was done to get a factor of safety on the line as high as could be practically obtained. The cables are supported from Locke porcelain suspension insulators, four units being used in series on suspensions and five units in series on dead ends.

Special care has also been taken in the design of this transmission line to provide for uninterrupted service under all conditions. The special features which will serve this purpose, in connection with those at the power plant mentioned under items *a* to *h*, are as follows:

i. The main line is built in duplicate, which always leaves the second line in service in case trouble develops on one line.

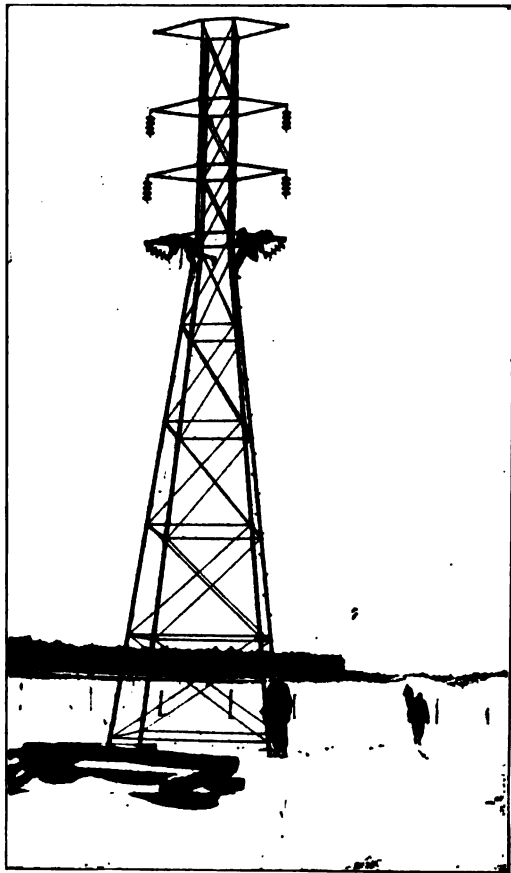


FIG. 10.—FOUR-POST TOWER.

j. The transmission line is protected over its entire distance by two overhead ground wires of No. 2 B. & S. copper-clad steel, strung 10 ft. 6 in. above the highest power cable.

k. Each transmission tower is thoroughly grounded by special ground plates below the concrete bases.

l. Each string of insulators is protected by special arcing rods arranged above and below the insulators to save them from damage by an accidental power arc.

m. The total line has been divided into a large number of separate sections, which may be easily and quickly disconnected by means of Dossert joints. This will greatly reduce the time for insulating and repairing a damaged section, if it should ever become necessary.

n. A private two-wire telephone line of No. 10 B. & S. copper-clad steel, supported on 6,000-volt insulators, has been installed on a wooden pole line, within a short distance of the transmission line, connecting the

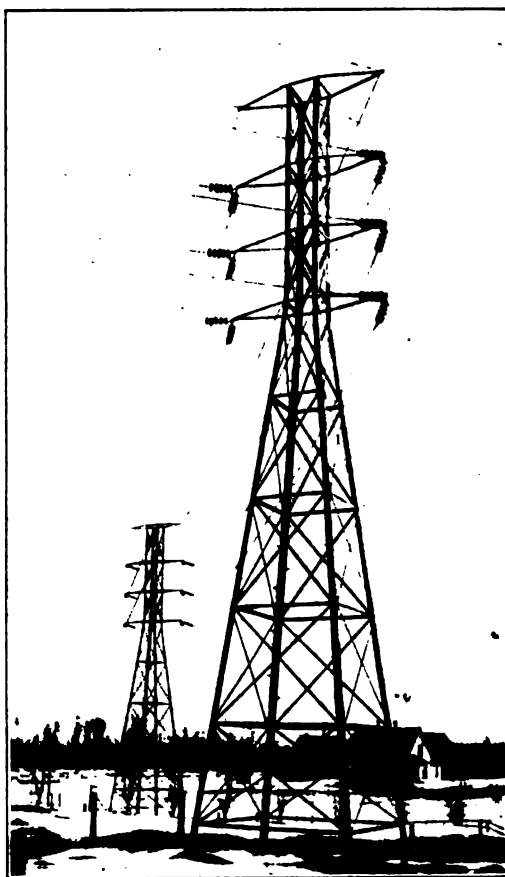


FIG. 11.—ANGLE TOWER.

main generating station with all the substations. The crews at the power house and all the substations are furthermore furnished with portable telephone sets which can be connected to any point of the telephone line. Each crew has also a portable acetylene search light at its disposal to enable it to patrol and repair the line at night.

With the above-named equipment, any trouble on the line can be quickly located, isolated, and repaired, considering also that the company

has at its disposal instant automobile service at all principal points along the line.

Auxiliary Steam Plant

The auxiliary steam plant is located just outside of the city limits of Iron River, which is central with respect to the present and possible future mining operations, and at the same time to the probable load center. The main line of the Chicago & Northwestern Railway through

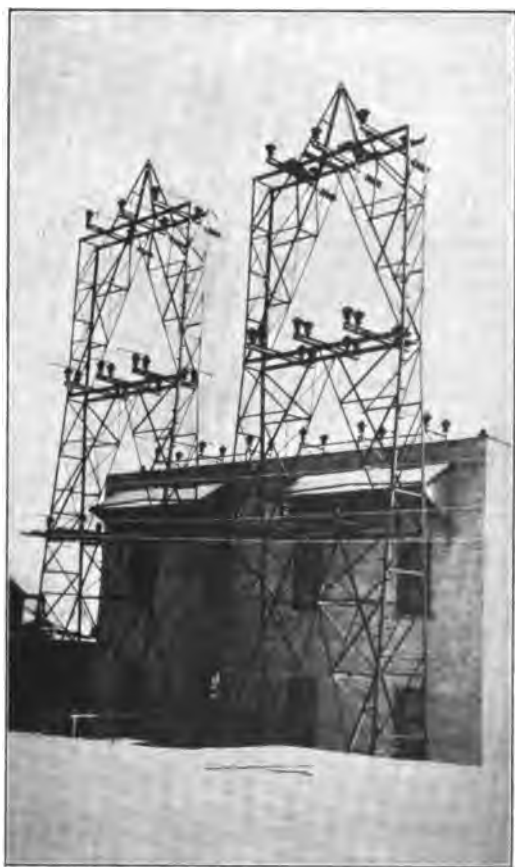


FIG. 12.—TOWERS AT IRON RIVER SUBSTATION.

this section adjoins the property on the east, providing excellent facilities for the delivery of coal. Just east of the railroad right-of-way is the Iron River, a stream having a drainage area of 60 square miles and an exceptionally uniform flow, thus assuring close at hand a sufficient supply of water for condensing and other purposes.

The permanent substation was completed first, and in connection therewith equipment, to provide enough steam capacity to take care of

power contracts in force before the completion of the hydraulic plant, was installed in a temporary wooden structure. Such portion of this steam equipment as was to be utilized in the permanent work was located on permanent foundations.

The first permanent installation, now completely finished and housed in concrete and brick buildings, consists of two Westinghouse turbo-generator units, one of 625 kw. and the other of 940 kw. capacity. The units generate three-phase, 60-cycle, 6,600-volt current. The turbines operate with 150 to 160 lb. steam pressure and 28-in. vacuum referred to sea level. On account of the elevation above sea level at Iron River, the indicated vacuum is about 27 in. of mercury.

The condensers are Westinghouse-LeBlanc with motor-driven pumps. Condensing water from the Iron River is taken from concrete wells located beside the condensers, the wells being connected with an intake in the Iron River by a 24-in. vitrified conduit.

For the initial steam-turbine installation, a boiler plant of three 150-h.p. long-furnace, internally fired boilers was provided, placed in a position out of the way of future boiler-plant construction. These boilers were selected on account of the facility of installation in the winter, very little bricking-up being required. They are provided with an extended breeching, connecting with a 60-in. by 75-ft. guyed steel stack set on a 25-ft. brick base, making the stack height 100 ft. above the grates.

The present permanent boiler equipment, now completed, consists of two 440-h.p. internally fired boilers, equipped with Hawley down-draft furnaces. These boilers are provided with an extended breeching connecting with a new reinforced-concrete chimney. The chimney has an inside diameter of 8 ft. at the top, and is about 154 ft. above the grates. The boilers first installed are still in their temporary boiler house and are connected to the new steam header for use as reserve.

In the boiler room two American steam pumps, one low service and one high service, are installed for boiler feed, together with one 1,800-h.p. Stillwell feed-water heater. The exhaust from the pumps and excitors is utilized in maintaining a feed-water temperature of 210° to 212°.

In the basement of the turbine room are the transformer oil-treating outfit, transformer cooling-water pumps, and general-service pumps, connected with an elevated tank in the boiler room for general station water supply.

The exciter units consist of one Westinghouse generator, driven by a vertical American Blower Co. engine, and one Westinghouse 10-kw. generator, driven by steam turbine. These units are on the main floor of the turbine room.

On the main floor of the turbine room, besides the main units and excitors, are the complete substation switchboard and duplicate sets of

15-kw. flywheel motor generators for control circuits and emergency lighting.

A 15-ton hand-operated traveling crane is provided for handling equipment in the turbine room.

The present plant is arranged and designed with a view to gaining high economy, and this is borne out by the results of operation during the first year. All coal used is weighed, which, with the records of electrical output, provides an index to the economy of operation. The stack temperature is obtained by an indicating and recording thermometer. A recording steam gauge and thermometer for reading all important steam, water, and oil temperatures is provided in addition to the usual steam and vacuum gauges.

Coal is dumped from cars, run out on a timber trestle, 100 ft. long and 20 ft. above the ground. This trestle is permanently constructed in a position parallel to the firing aisle and just outside the boiler room.

Main Substation

The main substation is at Iron River, Mich., and consists of a two-story brick building of fireproof construction, directly connecting with the company's auxiliary steam turbo-generator plant. It is arranged to receive two banks of transformers and the necessary switchboard and busbar equipment, all of which are now installed. There are six 420-kva. single-phase transformers of the oil-insulated water-cooled type. They are Y-connected on the high-tension side, receiving 66,000 volts between the leads, and delta-connected on the low-tension side, delivering 6,600 volts. Fig. 13 shows the wiring diagram of the substation.

The equipment for furnishing cooling water and that for filtering the transformer oil is a duplicate of that installed at the main generating station. The substation is designed to control two incoming 66,000-volt transmission lines, and four outgoing 6,600-volt distribution feeders; also, two banks of station light and power transformers, and all the auxiliary electrical equipment installed in the substation and adjoining steam turbo-generator plant.

The same type of protective equipment as is installed in the main generating station has also been placed at the substation, such as: Electrolytic lightning arresters on all high-tension and low-tension circuits; individual choke coils on all high-tension and low-tension circuits; high-tension station wiring of Swedish iron; special electrolytic cells for protection of current transformers; high-frequency arrester of European make; flywheel motor-generator sets in duplicate. In addition, the 66,000-volt oil switches for the incoming duplicate transmission line are equipped with reverse power relays.

A second high-tension substation has been built at Florence, Wis.,

10 miles from the hydraulic plant, and connecting to the main 66,000-volt transmission line. This substation supplies power to two iron mines near Florence, over a 6,600-volt line carried on steel towers. It contains three single-phase, oil-insulated self-cooled transformers, each of 150-kva. rating. These transformers are delta-connected on the high-tension and low-tension sides to allow of uninterrupted service in case of damage to one transformer by operating the remaining two units on open delta.

A third high-tension substation has been built and is now in operation at Alpha, Mich., supplying power to the Judson and Balkan mines.

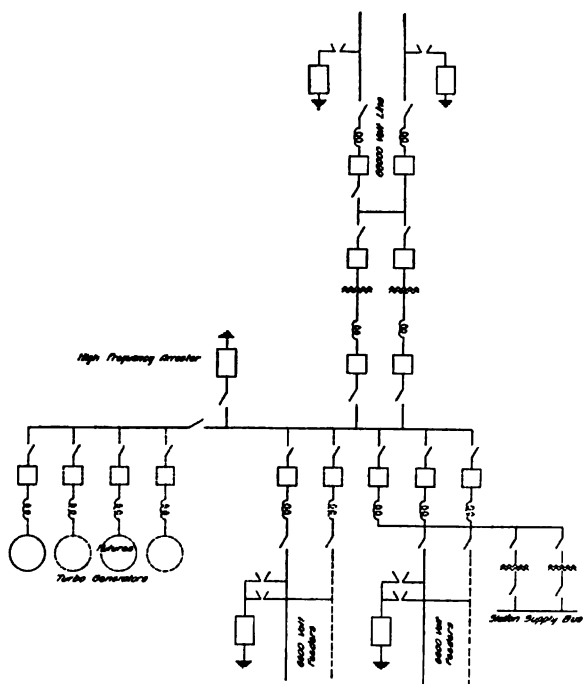


FIG. 13.—WIRING DIAGRAM, MAIN SUBSTATION AT IRON RIVER.

Arrangements are under way for furnishing the town of Alpha with electrical energy for general commercial lighting and power purposes. The equipment of this substation is the same as that at Florence, Wis., with the exception that the total transformer capacity amounts to 900 kw.

Secondary Distribution System

At present two three-phase, 6,600-volt feeder circuits are installed, distributing power to a number of mines around Iron River, Mich., and also furnishing energy to the local electric light and power plant. These feeders consist of the same size conductor as the main high-tension

transmission line (No. 0 B. & S. steel-reinforced aluminum strand), and are carried on 12,000-volt, pin-type insulators attached to steel towers set in concrete bases. The entire secondary distribution system, with the exception of one feeder which will be equipped later, is built in duplicate, thereby insuring uninterrupted service to the mines, even in the event of one circuit becoming totally disabled.

Fig. 14 illustrates the type of secondary tower that is used, the tower



FIG. 14.—TOWER OF SECONDARY DISTRIBUTION SYSTEM.

in this photograph being a tap-off tower. The main members in these towers are channel sections, cut longitudinally for about two-thirds their length and spread to form the base width of 4 ft. 6 in. Each half of the tower was shipped to the field riveted up, leaving the bracing between the two halves to be done in the field. The base dimension at right angles to the line is 8 ft. 6 in. The same method of constructing the anchorages was used as heretofore described for the towers in the main line.

Like the main high-tension transmission line, the secondary feeder lines have also been equipped with protective devices to eliminate disturbances and maintain uninterrupted service. The special protective features embodied in these secondary distribution lines are as follows:

The line is protected over its entire distance by an overhead ground wire of No. 2 B. & S. gauge copper-clad steel, strung 7 ft. above the highest power wire.

Each tower is thoroughly grounded by special ground plates below the concrete bases.

Each insulator is protected by a special arcing rod to save it from damage by an accidental power arc.

Practically all the secondary circuits supplying the mines are built in duplicate.

Secondary Substations

Power is transmitted to each mine at 6,600 volts and is there transformed down to 2,200 volts for general distribution around the mine. The equipment of each of these secondary substations is owned by the power company and consists of three single-phase, oil-insulated, self-cooled transformers, of a capacity corresponding to the particular mine load, with ample reserve capacity for a considerable increase of this load in future.

The equipment further comprises an automatic oil switch, a control panel with all necessary instruments, relays and measuring transformers, and a complete set of electrolytic lightning arresters and choke coils. The transformers are delta-connected on both the 6,600-volt and 2,200-volt sides to provide for continuity of service in case of damage to one transformer, allowing the operation of the remaining two units in open delta.

The secondary substations carry a typical mining load, consisting of air compressors, pumps, hoists, motor generators for underground haulage, crushers, blowers, shop motors, and lighting above and below surface.

Practically all the larger motors, such as compressor, pump, and hoist motors, are operated on 2,200 volts, which calls for only a comparatively small investment on the part of the mine owner in step-down transformers for the lighting and small motor load, thereby maintaining at the same time a high all-around plant efficiency.

In order to obtain the best possible voltage regulation over the entire system, all the large air compressors are equipped with synchronous motors, so adjusted as to give a range of power factor most advantageous for the maintenance of a steady voltage. The maximum variation in voltage in the distribution system is about 4 per cent., which is well

inside of the limits generally allowed by public service commissions in distribution systems for city lighting.

The economic and other advantages of electric drive for mining operations have been fully realized in the district served by the power company, which is evidenced not only by the number of new mines with electrical equipment throughout, but also by the mines formerly operated by steam and now changing to electric drive. The economy effected by electric over steam drive is more evident on the pumping operations. Three different companies supplied by electric power from this plant report savings of from 30 to 70 per cent. in the power cost for pumping.

A special arrangement has been made by the power company in regard to the large hoists, with their intermittent short-time heavy loads, which had an unfavorable influence upon the total load factor of the electric mine installation, on which the rates for service are partly based. The new arrangement provides that all the large hoisting motors be supplied over a separate set of meters, which eliminates the effect of the intermittent heavy hoisting load upon the meter determining the load factor, and thereby reduces the rate for power which would otherwise result.

The power taken by the hoist, and also the other motors, is not charged on the basis of the actual load factor obtained at the installation, but on the more favorable basis of a load factor obtained by that portion of the load exclusive of the hoist.

The writer is indebted to M. H. Collbohm, electrical engineer, and other members of the office force, for valuable assistance in the preparation of this paper.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Recent Developments in Coal Briquetting

BY CHARLES T. MALCOLMSON, CHICAGO, ILL.

(New York Meeting, February, 1915)

In the United States, improvements in methods of combustion have made possible the use of the smaller sizes of anthracite. This coal is now being reclaimed from the culm banks accumulated by the miners in the more prodigal years. To-day the freshly mined "slush" is segregated and stored against the time when, through briquetting or some other means, it may be utilized as a commercial fuel.

For many years Europe has achieved enviable results in the conservation of her small coal by briquetting. This profitable industry, first placed on a commercial basis in France in 1842, has grown to such magnitude that the production during 1913 exceeded 36,000,000 short tons. Briquets produced in Great Britain are used principally for steam purposes and the greater part of the output is either exported or taken by the Admiralty. On the Continent the briquets are sold in all branches of the coal trade. The French railways consume from 16 to 40 per cent. of their fuel in briquet form, and in Germany the lignite briquets (of which 20,000,000 tons were produced in 1913) are as familiar and interesting to the American traveler as the huge porcelain stoves in which they are burned.

It is not strange, therefore, that aggressive and far-sighted promoters heralded this means of converting our ubiquitous coal waste into revenue. To the superficial student the briquetting process seemed so simple as to make success easy. It is for this reason principally that the path of the industry in America has been strewn with failures. The rash enthusiast had either not worked out the mechanical problems involved or had failed to grasp the local conditions under which the briquets must be made and sold. It may be confidently stated, however, that in spite of the many vicissitudes through which it has passed, coal briquetting is now an accomplished economic success in this country, and as a profitable adjunct to the coal industry it has come to stay.

Several valuable contributions relating to coal briquetting are found in the *Transactions*. Charles Dorrance, Jr., has sketched briefly the history of the anthracite briquetting plants from the time of Loiseau's pioneer plant down to that of the Lehigh Coal & Navigation Co. at Lansford. Since then this phase of the industry has remained practically at

a standstill in the East; in fact, several of the plants then in operation have ceased to exist.

The experience of the past few years points to the conclusion that the anthracite product as now manufactured is not what the consumer wants. Briquets marketed on the Eastern seaboard are nearly all made with coal-tar pitch as a binder, following the European practice, and are sold for domestic purposes, at from \$1 to \$1.50 below the price of chestnut coal. While the culm may not contain more ash than is allowed by the government specification for prepared sizes of anthracite, the ash is thoroughly distributed through the briquet, and this feature, combined with conditions under which the fuel is consumed, causes the binder to distill off at lower temperatures than the coal burns, thus producing some soot and smoke. While this defect is not great, the difference in price does not seem to compensate for it. Other binders which will eliminate this objectionable feature are now available.

There is, however, an indifference on the part of the large anthracite operators which is not encouraging. It has been stated that every means is being employed to increase the production of anthracite, and that the demand is considerably in advance of the supply, and is growing steadily. However, the fact that coal is being stored at the present time does not bear out these statements. As one operator puts it, so long as there is a surplus of the high-priced sizes, no effort will be made to deplete the market for these sizes by the introduction of briquets from the low-priced sizes.

On the other hand, it is encouraging to observe that the plant of the Scranton Anthracite Briquette Co. at Dickson City, Pa., is selling its entire output of 350 tons per day for locomotive consumption, and that these pitch-made briquets produce no deleterious effect. The briquets are mixed with bituminous coal and are proving a successful engine fuel.

It is not strange that one of the earliest practical demonstrations of the briquetting industry took place on the Pacific Coast. Since California produced only inferior grades of lignite, high-priced Welsh bituminous and anthracite coals, Australian and some Japanese coal were imported in considerable quantities up to the time that the better grades of native coals from the Rocky Mountain States and Washington entered this market. Robert Schorr has already presented to this Institute a description of the plants built by him at Stockton and Oakland. The latter plant is still intact, but the supply of screenings produced in the handling of imported coals is now so small as to render briquetting no longer profitable.

In the meanwhile the most important progress in the briquetting of bituminous coals has been made in the Middle Western States. I will outline briefly the plants which mark this development.

The greatest single impetus to the industry was given by the U. S.

Geological Survey through its experiments at St. Louis and Norfolk from 1904 to 1907. During this period the Semet-Solvay Co. erected a plant at Detroit to utilize its accumulations of coke breeze. Results at this plant demonstrated the necessity of equipment designed to meet American conditions. The subject has been well covered in a paper by W. H. Blauvelt before this Institute.

As a direct result of the government work, the Western Coalette Fuel Co. built a plant at Kansas City, Mo., using an 8-ton per hour Renfrow press of the plunger type producing 12-oz. briquets. This plant was unsuccessful because the mechanical problems had not been given proper consideration.

Profiting by this experience, the Renfrow company built another press having a capacity of 8 to 9 tons per hour, and making briquets weighing 13 oz. This press was installed in the Norfolk plant of the Survey and made the briquets which were tested on the Eastern railroads and for the Navy Department. Upon concluding the Norfolk tests the machine was sold to the Rock Island Coal Mining Co. and installed at Hartshorne, Okla.

The Hartshorne plant operated about three years on a successful commercial basis, briquetting the bituminous slack mined by the company. The product was marketed under the trade name of "Carbonets." When this project was undertaken, slack coal was accumulating rapidly, as the Mexican market for McAlester coke had ceased to exist, and the coke ovens which had consumed this slack were shut down. Later, however, a prolonged strike in this region, together with the decrease in Texas oil production, and the opening of new markets in the Southwest, soon made the price of slack prohibitive for briquetting.

It must be borne in mind that the most important commercial feature in the marketing of briquets is the margin between the selling price of the fine coal to be briquetted, and that of the prepared coal which the briquets meet in competition. It is principally because of the variableness of this margin that the coal-briquetting industry has not been successful in every part of the United States where it has been tried.

The Hartshorne plant demonstrated the fact that the briquets could be sold when a uniform product was manufactured. Briquets weighing from 8 to 16 oz. met the market requirements in the Middle West as a competitor for 3-in. bituminous or anthracite coal. Simpler and more efficient machinery, however, was required. In 1909, the Standard Briquette Fuel Co. built a plant at Kansas City, Mo., to try out a 10 ton-per-hour plunger press, following the German and English design but modified to meet American conditions as to size of briquets. The plant was an improvement over former designs, but after two years of costly experimentation the press was replaced by another of the same tonnage. Even after the second press was perfected mechanically, during a period of

two years, its operating cost was too great to make briquetting profitable on the margin obtaining at Kansas City.

About this time a plant of somewhat similar design and using a Renfrow press was built by the Detroit Coal & Fuel Co. It is still in operation, producing 12-oz. briquets out of Pocahontas coal at the rate of 8 tons per hour.

The same necessity for conservation which found a use for the hitherto waste coal in the culm banks of eastern Pennsylvania, found a market for the fine coal in the great Mississippi Valley region. This market was created by the development of the mechanical stoker, and as a result the prices of slack and lump coal were more nearly equalized. Hence it transpired that the future of bituminous briquets must be limited to domestic use, and that the industry could be made profitable only by developing highly efficient machinery of large tonnage. The railroads



FIG. 1.—BRIQUETTING PLANT OF BERWIND FUEL CO., SUPERIOR, WIS.

and large coal operators could be interested only on this basis. The Rutledge press was designed to meet these conditions.

The St. Louis Briquette Machine Co. began in 1909 the construction of a plant at Livingston, Ill., to be used in perfecting the Rutledge press, and to learn something of the value of Illinois briquets. The plant had a capacity of 32 tons per hour of 16-oz. briquets. In 1911 the press had been sufficiently developed to warrant the Berwind Fuel Co. in contracting for the construction of a plant on its dock at Superior, Wis. (Fig. 1), which was completed the following year.

A description of the Berwind plant has been published in the technical press, and need not be repeated here; however, the reasons for establishing



FIG. 2.—BERWIND BRIQUETTING PLANT, SUPERIOR, WIS.



FIG. 3.—BRIQUET-HANDLING EQUIPMENT, BERWIND PLANT, SUPERIOR, WIS.

the plant at Superior may be interesting in this connection. In 1902 approximately 1,000,000 tons of anthracite and less than 10,000 tons of Pocahontas coal had been received at the ports of Duluth and Superior. In 1905 the demand for Pocahontas coal had practically ceased and this condition continued until the Berwind dock was built in 1907. In 1911 these receipts were increased to 1,365,000 tons of anthracite and 520,000 tons of Pocahontas and other "smokeless" coals.

Pocahontas coal is essentially a domestic fuel, so that, in order to compete successfully with anthracite, preparation of some kind is necessary. On account of the friability of the "smokeless" coals, the degradation in handling from the tipples in West Virginia to the docks at Superior or Duluth is enormous. It soon became apparent that with over 65



FIG. 4.—STANDARD BRIQUETTE FUEL CO.'S PLANT.

per cent. of the "run of pile" coal passing through a $\frac{3}{4}$ -in. bar screen, either the market for lump and egg sizes would be curtailed or screenings must be moved at a sacrifice in price. Difficulties were also encountered in maintaining the proper demand for the various sizes so that they could be moved as fast as they were prepared and thus avoid the cost and breakage of storing and rehandling. Even with the best preparation and the most careful handling there was always a large percentage of slack in the "egg" and "lump" which the consumer put into his coal bin.

These were the conditions which made the installation of the Berwind plant opportune and profitable. Within a year after the operations were begun a second complete unit was installed, bringing the rated capacity of this plant up to 80 tons of 13-oz. briquets per hour, and making it by

far the largest plant in the country. Thus the briquetting industry at the head of the Lakes was from the outset placed on a sound economic basis (Figs. 2 and 3).

Adequate and attractive advertising brought the merits of briquets before the people of Minnesota and the Dakotas, the principal market for this product. It took some time to convince the dealers that briquets

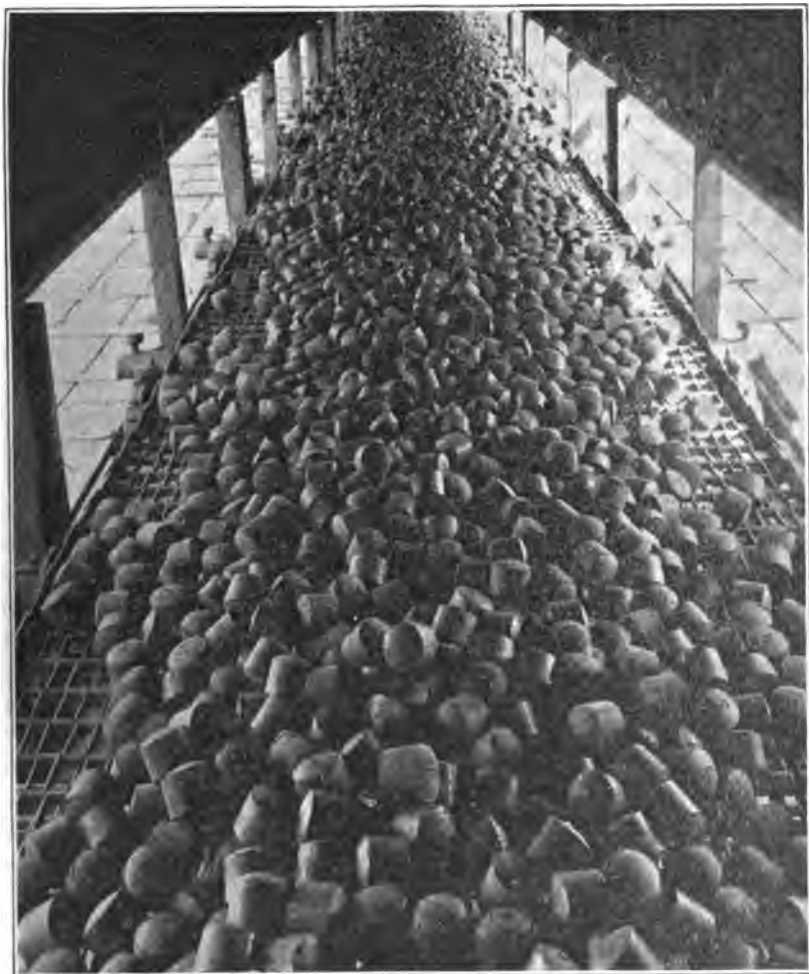


FIG. 5.—BRIQUETS MADE AT STANDARD PLANT.

were cleaner and more economical than the raw coal which they had been buying. The demand for this fuel, however, has steadily increased, which is the best evidence of the continued success of this enterprise. If Pocahontas briquets are not displacing anthracite they are supplying

the demand due to the natural expansion of the country. At the present time briquets are being sold at the same prices as the prepared Pocahontas coal; in this way the market for both products is maintained.

The Standard Briquette Fuel Co. now felt justified in having its plant entirely re-designed and enlarged to accommodate a Rutledge press producing 30 tons of 10-oz. briquets per hour (Figs. 4 and 5). Arkansas semi-anthracite coal is now briquetted with 6 per cent. coke-oven pitch and the briquets are sold in Kansas City and contiguous territory. This is the second season for the new plant, and it is now, for the first time, operating on a successful basis both mechanically and commercially.

Before passing to a description of the plant which forms the basis of this paper, it will not be amiss to mention the Stott Briquette Co.'s plant at Superior, Wis., built in 1910. After several unsuccessful attempts to exploit anthracite briquets, the company decided this year to manufacture a product consisting principally of Pocahontas coal. A modified Belgian roll press built by the Mashek Engineering Co. is used, producing 2-oz. pillow-shaped briquets at the rate of 12 tons per hour. The residue from petroleum is used as a binder.

In all coal briquetting the binder is a factor of prime importance. Coal-tar pitch has been generally recognized in Europe as the logical binder for coal briquets. It was adopted in this country with the industry. There are many admirable reasons for its use. It is cheap, easy to transport and to handle in the manufacturing process, is waterproof, acts as a preservative to the coal, and produces coke from an otherwise non-coking coal. It is a coal product, and if properly incorporated with the coal becomes a part of it during combustion. When hard pitch is used in dry form, the briquets thus made with bituminous coal produce less smoke than the raw coal. It has one serious defect: it will not make a smokeless fuel out of anthracite culm.

To remedy this evil, binders of vegetable origin have been much exploited. They are, however, expensive, difficult to handle, and do not make waterproof briquets. Two small plants in the country are using vegetable binders of which starch is the principal ingredient, and one plant uses oil emulsified with starch. In all cases the briquets must be baked before shipment and the product is irregular in quality. The processes can hardly be said to be commercially successful.

Another excellent binder is asphalt obtained from the distillation of petroleum. It has many of the qualities of coal-tar pitch; the difficulties in its use will be referred to later.

There is a steady and growing demand in the Pacific Coast States and in British Columbia for coals mined in that region, although the higher-grade coals from Wyoming and other Rocky Mountain States also reach this market. Until a few years ago the demand for steam and

domestic coals was about equal, so that the coal operators could market their prepared coals without throwing an undue surplus of any one size on the market. The enormous increase in the production of petroleum in California, however, necessitated the creation of an adequate market, and the price of fuel oil was therefore reduced in order to move this production. For industrial plants and railroad purposes steam coal was rapidly displaced by the economically superior oil, so that by the end of 1913 practically all the furnaces in the locomotives, steamships, and large steam plants were transformed to burn oil.

The Pacific Coast Coal Co., as the largest producer of coal on the Pacific Coast, early realized what this condition would eventually mean, and began to search for some means to convert its now unmarketable fine coal into a profitable domestic fuel. The late James Andersen, Chief Engineer of the company, was delegated to make this investigation, after he had visited many briquetting plants in the United States and Europe covering a period of three years, the Malcolmson Briquet Engineering Co., of Chicago, was awarded a contract to design and build a complete coal-briquetting plant.

At Black Diamond, Franklin, and Burnett, a good grade of bituminous coal, relatively low in ash, is mined by the Pacific Coast Coal Co.; the New Castle coal, however, falls within the government classification of sub-bituminous coals. While the larger sizes furnish an excellent domestic fuel, for which there is a ready sale, the New Castle fine coal was the first to be displaced by the introduction of oil. This coal will not coke, but this objectionable feature is overcome by the admixture of a small percentage of the company's South Prairie screenings, which produce a strong coke. The result is a highly desirable domestic fuel, meeting all the requirements of a merchantable briquet.

The coal-briquetting plant of the Pacific Coast Coal Co. is located near Seattle, Wash., close to the shore at the southern end of Lake Washington. It is served by the New Castle branch of the Columbia & Puget Sound Railroad, from which a spur passes the plant on its way to the coal piers at this end of the lake.

No more ideal location for a manufacturing plant can be found. A virgin forest was cleared in preparing the site. An abundance of water is pumped from the lake at small expense. An equable and salubrious climate lends itself to the efficient and continuous operation of the plant (Fig. 6).

The raw coal is delivered to the plant in dump-bottom cars, and is unloaded through a track hopper alongside the raw-coal bin. A gravity-discharge elevator, into which the coal is delivered from the track hopper by a reciprocating feeder, elevates the coal to an 800-ton raw-coal storage bin, of wooden construction and divided into two compartments.

Two feeders deliver the coal from these compartments into two flight conveyors. These feeders are designed for close regulation so that the requirements of the plant can be adjusted at this point. The feeders are driven through friction clutches by an electric motor.

The raw-coal conveyors travel horizontally and terminate at the intake chutes of the driers. Between the driers and the raw-coal bin a Williams dustless crusher is installed. Gates and chutes permit the delivery of coal from either raw-coal conveyor to the crusher, from which it is discharged into a continuous-bucket elevator and is again elevated to the raw-coal conveyors.

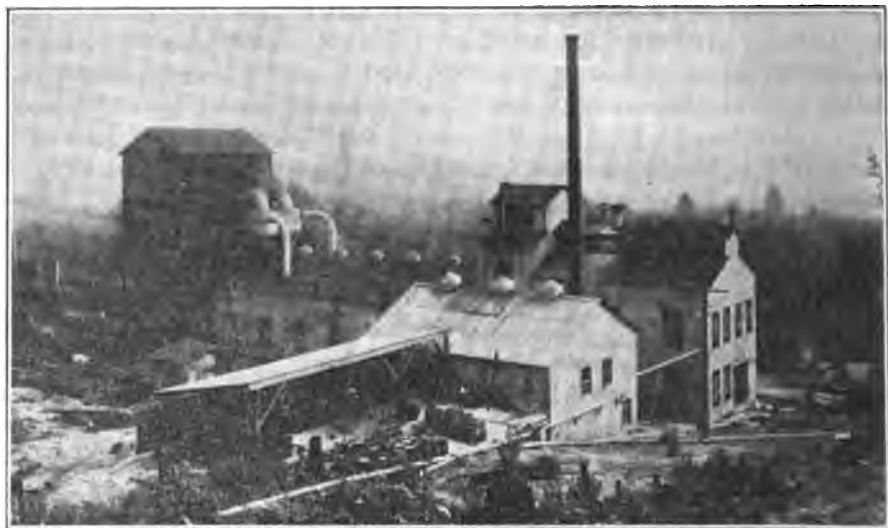


FIG. 6.—BRIQUETTING PLANT OF PACIFIC COAST COAL CO., NEAR SEATTLE, WASH

If the coal is very wet, it may be passed directly through either drier to a flight conveyor installed between the driers, which returns it to the crusher. The necessary part of the surface moisture is eliminated in the first drier, and after crushing the coal is completely dried and heated in the second drier.

This duplicate arrangement of driers and conveying machinery permits the blending of two or more coals to improve the quality of the mixture, and insures uniformity of the dried product.

Two Ruggles-Coles A-14 driers of improved design, with shell of special length, are installed in the drier building (Fig. 7). Each of these driers is direct-driven by an electric motor of sufficient size to pick up the load under all conditions. The exhaust fans are driven by variable-speed motors through silent-chain drives, and the exhaust gases pass through

cyclone separators located above the roof of the drier building. The dust from these separators is returned to the raw-coal conveyors.

Fuel oil furnishes the heat for drying the coal, and is introduced into the driers through Billow furnaces built by the National Supply Co. of

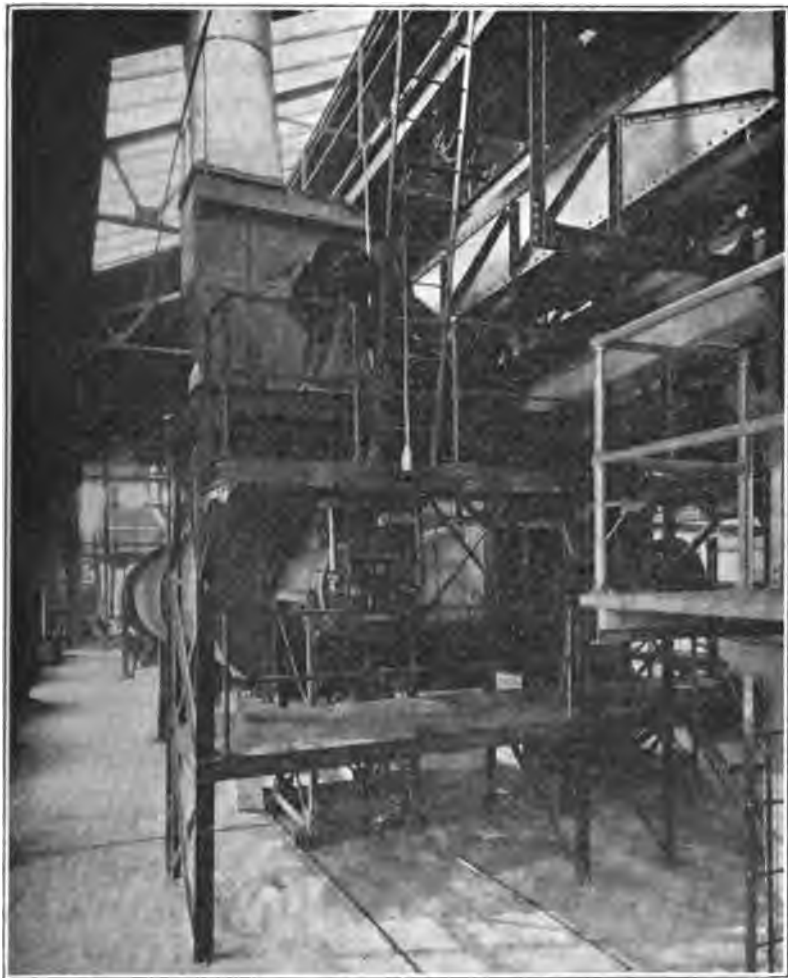


FIG. 7.—DRIER BUILDING, PACIFIC COAST COAL CO.

Chicago. A General Electric centrifugal air compressor furnishes air for atomizing the oil, and perfect combustion is thereby obtained. These are the most satisfactory furnaces yet installed by us for drying coal, in that they permit very close regulation and a uniform temperature of the dried coal at the discharge end. The furnaces are carried on structural-steel

frames supported on wheels, so they can be removed without much expense whenever it is necessary to enter the driers for repairs.

The dry coal is taken from the driers by means of a continuous-bucket elevator, to a small dry-coal bin, located directly under the roof of the machine building. All of the equipment, from the raw-coal feeders to and including the dry-coal bin, is of steel construction and absolutely dust tight. This feature increases the efficiency of the drying system by preventing the ingress of cool air, as well as insuring a dust-free atmosphere in the buildings.

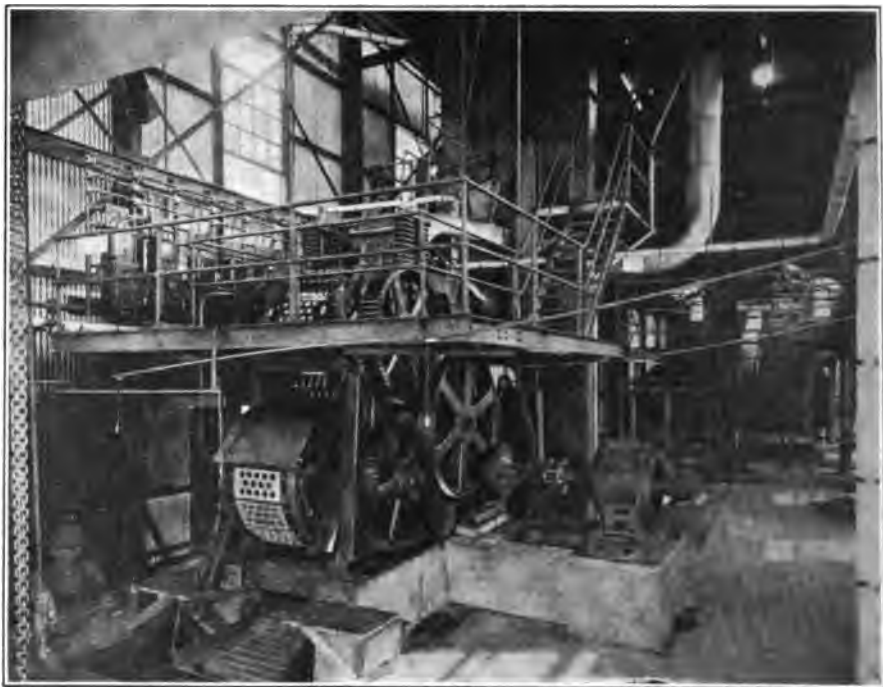


FIG. 8.—RUTLEDGE BRIQUETTING PRESS AND FLUXER.

Underneath the dry-coal bin an apron feeder is located, which serves to regulate the quantity of coal required by the briquetting press, and also to deliver this coal to the preliminary mixing equipment.

Liquid binder is maintained at constant pressure and uniform temperature in a small storage tank, located directly above the mixing equipment, and is introduced through a Forman steam-jacketed regulating valve into the mixer containing the dry coal. A uniform flow of binder is insured by this equipment and the proportioning of binder and coal is under the immediate control of the operator on the charging platform. Taylor index thermometers located here indicate the temperatures of

coal and binder; and the dust-collecting system installed above this equipment removes all dust caused by the handling of the dry coal.

After the coal and binder have been subjected to a preliminary mixing they pass in a uniform stream to the Rutledge fluxer (Fig. 8). This piece of special machinery is the result of long experimentation in the development of equipment for handling and mixing large quantities of coal and binder at relatively high temperatures. A thorough blending or "fluxing" of the coal and binder is of prime importance in obtaining satisfactory and uniform briquets at low cost. A simple mechanical mixing of the raw materials will not achieve the desired end; a thorough coating of the coal particles with a thin film of binder, so intimately associated as to approach a chemical union, is the result striven for and obtained by this efficient equipment.

The Rutledge fluxer consists essentially of a large chamber and two partly inclosed superimposed sections, where superheated steam is introduced into the mixture through a carefully worked out piping system. These horizontal shafts are provided with a series of paddles so arranged as to agitate and thoroughly flux the coal and binder as it passes downward through the successive chambers. The temperature of the agglomerated mass is raised at a uniform rate and with greater economy in the successive stages owing to the partly inclosed chambers. The fluxer is operated through a train of gears driven by an electric motor through a silent-chain drive and friction clutch.

The use of asphalt obtained from the distillation of petroleum as a briquet binder has been perfected through a process invented by Robert Schorr. This process was developed in the operation of several small plants which were built some years ago in California under his direction. The briquetting of coal with this binder has never before been attempted on such a large scale. Considerable time was spent in adapting the Schorr process to the conditions obtaining at the Seattle plant. The results finally produced are an indication of the excellence of this process and prove it indispensable in the manufacture of briquets using asphalt as a binder.

The Rutledge briquetting press installed in this plant is of the same type as those used in the plant of the Berwind Fuel Co. at Superior, Wis., except that it manufactures a smaller briquet and incorporates the improvements which grew out of the operations at Superior (Fig. 9).

The Rutledge press has been designed to meet American conditions by producing a large output of relatively small briquets under high pressure with a small percentage of binder. The handling of 30 to 40 tons of coal per hour through a manufacturing plant involves problems which are not anticipated by the experience in briquetting 8 or 10 tons per hour, the usual output of the other machines on the market.

The Rutledge press is of the continuous-mold type. It consists

essentially of a chain of molds or die plates, passing under a feeding hopper and between two revolving drums, upon which punch rams are carried. Each die plate contains 14 dies. Twelve punch rams are mounted on the upper and 12 on the lower drum; each punch ram supports 14 punches. Two pilot punches on each punch ram engage the

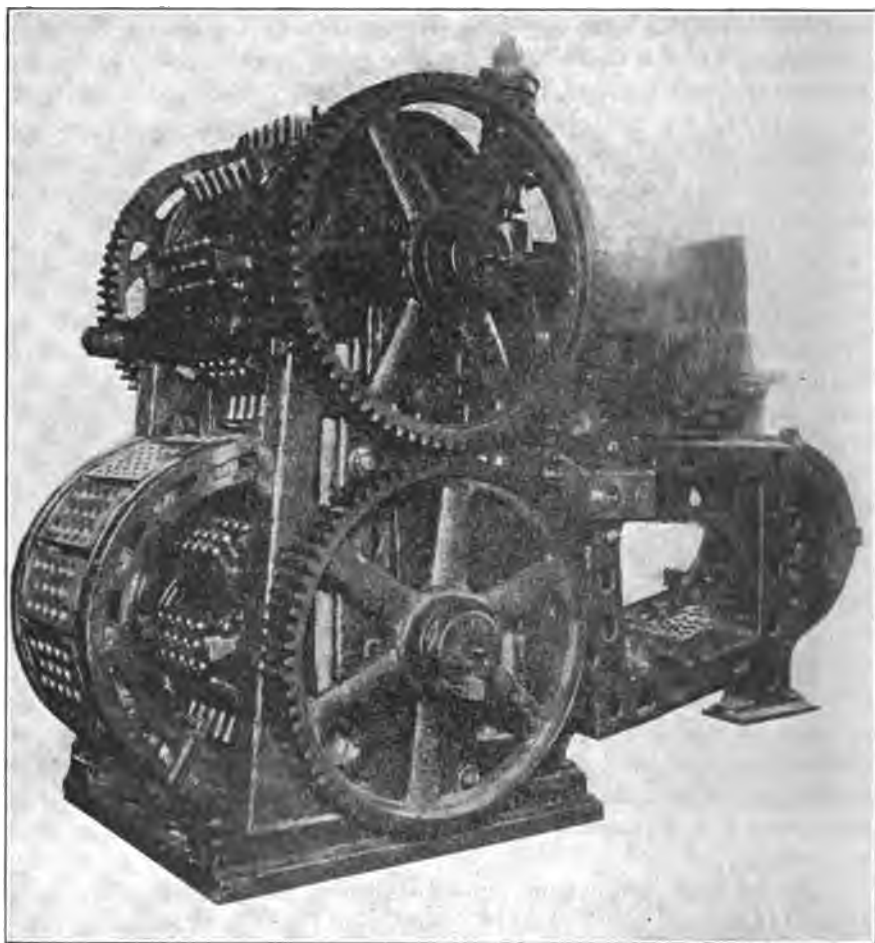


FIG. 9.—RUTLEDGE BRIQUETTING PRESS.

die plates in advance of the entrance of the punches into the dies; these pilot punches not only serve to move the dies forward, but insure perfect alignment of the dies and punches. The lower punches enter the dies again at the lowest point of their travel, and eject the briquets.

The punch rams oscillate on seats in the drums; the movement is produced by arms, cast on the rams, traveling in fixed cam tracks. This

movement is so timed with relation to the angular displacement of the punch rams about the main shafts upon which they are mounted, that the punches enter the dies and compress the briquets with a straight-line motion.

The bearings of the upper drum are set in vertical guides, and operate against heavy helical springs. These springs are set to move upward when the pressure of the punches on the briquets exceeds 4,000 lb. per square inch. Under normal working conditions the springs compress slightly at every compression of the punches. The briquets are cylindrical in shape, with spherical ends, and weigh $10\frac{1}{2}$ oz. A diamond stamped on one end identifies the "Black Diamond" briquet. For the briquetting of New Castle coal, punch tips containing the letter "N" have been provided. The press is turning out 32 to 33 tons of briquets per hour.

The Rutledge press and fluxer are built entirely of steel, with the exception of the bearings, which are of special bronze alloy. The parts receiving the most wear, such as the die bushings, cam track, and punch-ram guide rollers, are of chrome-nickel steel, heat treated, or manganese steel, and are easily replaceable.

The briquets are discharged from the press on to a perforated chute which delivers them to the cooling conveyor (Fig. 10). All of the "fines" from the press and any damaged or inferior briquets pass through this chute into a "refuse" flight conveyor. This conveyor passes underneath the press and delivers its charge into the elevator which returns it to the fluxer.

The cooling conveyor is composed entirely of link-belt malleable chain. The head shaft carrying the sprockets is driven through the necessary gears by an electric motor. This conveyor passes horizontally below the floor line, from the front of the press to the outside of the machine building; whence it travels a short distance on a slight incline following the rise of the ground; and again horizontally on a wooden trestle until it terminates at the lower floor of the briquet loading pocket. The speed of the conveyor is fixed so that it may be loaded uniformly and the briquets may have the greatest opportunity for cooling. The surface cooling of the briquets is also assisted by the use of water in a fine spray delivered from nozzles installed along the course of and above the conveyor.

The briquet loading pocket is a wooden structure of small capacity. It is not intended for storage purposes, except to receive the briquets during the shifting of cars. While cars are being loaded the briquets are drawn from the pocket through duplex undercut gates into a gravity-discharge bucket elevator, which also serves to convey the briquets from the end of the cooling conveyor to the loading chute, or elevate them to the briquet pocket, as desired.

The asphalt used as binder comes from California, and is received in barrels on a spur of the main line of the New Castle branch at an elevation of about 80 ft. above the foundation line of the briquet plant. The barrels are skidded on a chute to a small storage yard adjacent to the boiler and melting-kettle house. Here they are stripped as required, and the asphalt is charged by hand into the rear end of the kettles.

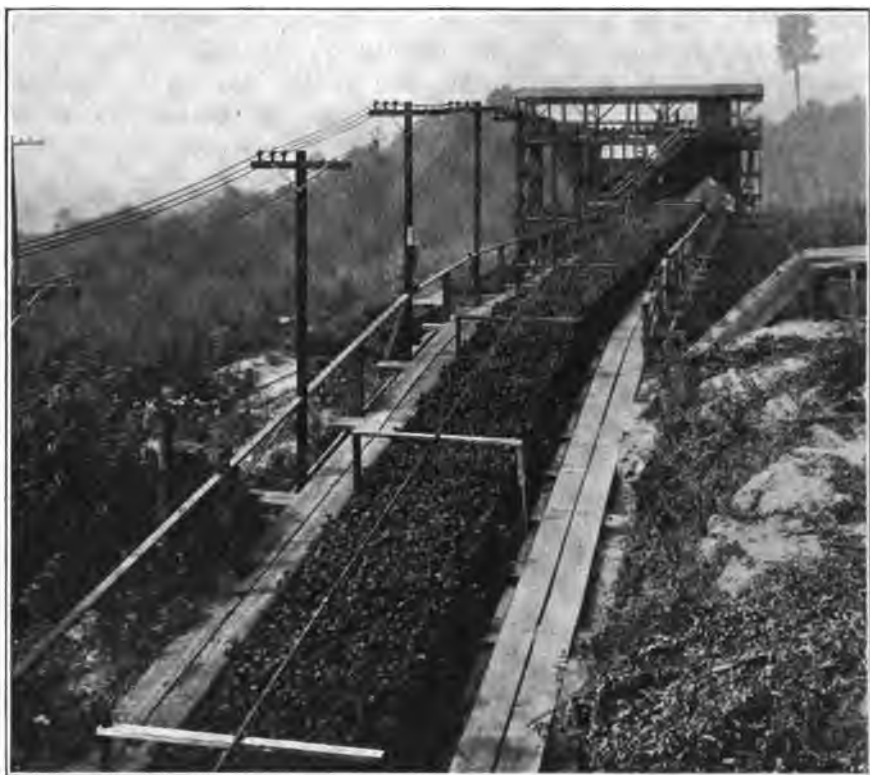


FIG. 10.—COOLING CONVEYOR.

There are two melting kettles, of the same size and design, built with shells of heavy sheet steel hung from structural-steel frames. Fuel oil furnishes the heat for melting the asphalt. Care has been taken in the design of the furnaces to distribute the heat uniformly, and prevent the usual difficulties due to intense local heating. These furnaces are also designed for the greatest fuel economy, in that the gases from one furnace may be passed through the other furnace before reaching the stack. Provision is also made to heat the incoming air with the exhaust gases. Billow oil furnaces, similar to the drier furnaces, are used. A stirring device directly driven by a vertical motor is placed in one kettle. The

usual procedure is to melt the asphalt in one kettle and retain the other full of melted asphalt at the proper temperature, as a reservoir, from which it passes to the pump.

The intention of the Pacific Coast Coal Co. is to erect eventually a complete distilling plant and manufacture its own binder. It is for the purpose of consuming the distillates from this process that oil-burning instead of coal furnaces have been installed.

In front of the melting kettles a Kinney rotary steam-jacketed pump is installed, together with an electric motor which operates the pump through a silent-chain drive. The piping on the front of the kettles is so arranged that the asphalt can be drawn from the bottom or side of either kettle to the pump. The hot asphalt is pumped directly to the small reservoir above the charging platform of the briquet-machine building, and any overflow from this reservoir is returned to the melting kettles. All of the asphalt pipes are incased in asbestos covering and are steam-jacketed to insure continuous operation.

The fuel oil is delivered to the briquet plant in tank cars and discharged into a reservoir located beside the upper unloading track, from which it passes to the Billow oil-pumping system in the boiler and melting-kettle house. This equipment strains and heats the oil and pumps it to the various points of consumption under uniform pressure. The system is automatically controlled; and to its efficiency may be largely attributed the successful operation of the oil-burning equipment. All of the oil pipes are insulated and heat-protected.

One 72 by 16 horizontal return-tubular Wickes boiler furnishes steam for the fluxer, the operation of the steam pumps, and if necessary the oil burners. Oil is also used for boiler fuel.

The main buildings are built entirely of structural steel with corrugated sheet-steel siding and asbestos-protected metal roofs. Steel-sash ventilators with mechanically operating devices are used in the machine building, and star ventilators in the drier building and boiler house. All the machinery supports and stairways are of structural steel, and the main platforms around the press and drier are of checkered plate. The only wood used in the main buildings is on some of the upper platforms and the windows and doors.

All the conveying and elevating machinery was manufactured by the Link-Belt Co.; it is of ample capacity and of very heavy construction, including steel gears, sprockets, and driving chain. As far as possible, machinery of standard design has been utilized or modified to meet the special requirements of the plant. This reduces the cost of repair parts and makes them readily obtainable.

An independently fired Foster superheater built by the Power Specialty Co. is installed in the corner of the machine building opposite the

briquetting press. A small Billow oil-burning furnace furnishes the necessary heat.

General Electric electrical equipment is used throughout. Slow-speed motors and individual drives are installed, and in most cases direct connected. Wherever the necessary reduction in speed is great, silent-chain drives are used.

The control of all electrical equipment is concentrated at two points in the drier and machine buildings. Pipe boards, built by the General Devices & Equipment Co., carry all of the control apparatus for these motors. Two panel boards distribute the current for the electric lighting system. All electric wires are carried in steel conduit, and the latest approved type of safety devices is used.

Three-phase, 60-cycle alternating current generated by the hydro-electric station at Snoqualmie Falls is purchased of the Puget Sound Traction, Light & Power Co., and is delivered at 2,300 volts to the transformer station at the briquet plant. It is here reduced to 440 volts for power and 110 volts for lighting service. A pole line was built by the Pacific Coast Coal Co. to transmit this current from the substation at Renton.

A complete water system has been put in by the company to protect the auxiliary buildings of the plant which are of wooden construction. A small centrifugal pump, driven by an electric motor with remote control, pumps the water from Lake Washington to a small reservoir located directly in front of the main building. Here a high-pressure steam-driven Underwriters' pump is installed to pump the water to a 50,000-gal. tank on the hill beside the oil reservoir. Hydrants are located at all desirable points.

A description of this necessary and valuable adjunct to the coal properties of the Pacific Coast Coal Co. would not be complete without reference to the assistance rendered by N. D. Moore, Chief Engineer of the Pacific Coast company, and Ralph Galt, now Superintendent of the plant, under whose direction the building and installation work was carried on.

The plant has been in operation since the latter part of August, turning out merchantable briquets, which have been distributed to the local trade and neighboring cities in order to satisfy as far as possible the demands for this fuel. Advices from the sales department indicate that the briquets are everywhere being received with satisfaction. They are sold in competition with the best grades of bituminous coal produced in the Pacific Coast States and Canada. The fact that they are sold in Victoria and other British Columbia cities in competition with the high-grade bituminous coals produced on Vancouver Island is the best evidence of the superiority of briquets over coal in its raw state.

The Pacific Coast Coal Co. has made the necessary preparations to double the capacity of this plant. The second unit and the necessary auxiliary equipment will be installed as soon as it becomes evident that the permanent demand for briquets has reached the maximum output of the present plant.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Structure and Hysteresis Loss in Medium-Carbon Steel

BY F. C. LANGENBERG,* CAMBRIDGE, MASS., AND R. G. WEBBER,† ATHENS, OHIO

(New York Meeting, February, 1915)

DURING the course of some magnetic investigations which the authors have under way, six bars of 0.43-carbon steel were tested, a permeameter designed after the Hopkinson yoke type being used. The results obtained were of some interest and although far from complete serve at least to show the necessity for a careful investigation of the previous history of the samples under study as well as the chemical composition.

A steel of given chemical composition may assume a great variety of structures, depending on its treatment. Professor Sauveur has devised a very clear and effective means of illustrating this point. The accompanying microphotographs show some of the possible structures assumed by a 0.30 carbon steel (Fig. 1).

It is now a well-established fact that a structural change is accompanied by changes in the tensile strength, elastic limit, hardness, etc., and it seems reasonable to assume also, a change in the magnetic properties.

Six bars of a 0.43-carbon steel were used, all the bars being taken from the same rod and treated as follows:

- Bar 1. Heated to 1,100° and cooled in the furnace.
- Bar 2. Heated to 1,000° and cooled in the furnace.
- Bar 3. Heated to 900° and cooled in the furnace.
- Bar 4. Heated to 1,000° and cooled in air.
- Bar 5. Heated to 900° and cooled in air.
- Bar 6. Untreated.

The bars were then turned to 13 mm. diameter and tested by the Hopkinson yoke method. After testing, three specimens were taken from each bar, one from each end and one from the middle. The homogeneity of the bars was tested in this manner and in no instance could any difference in structure be detected between the three specimens.¹

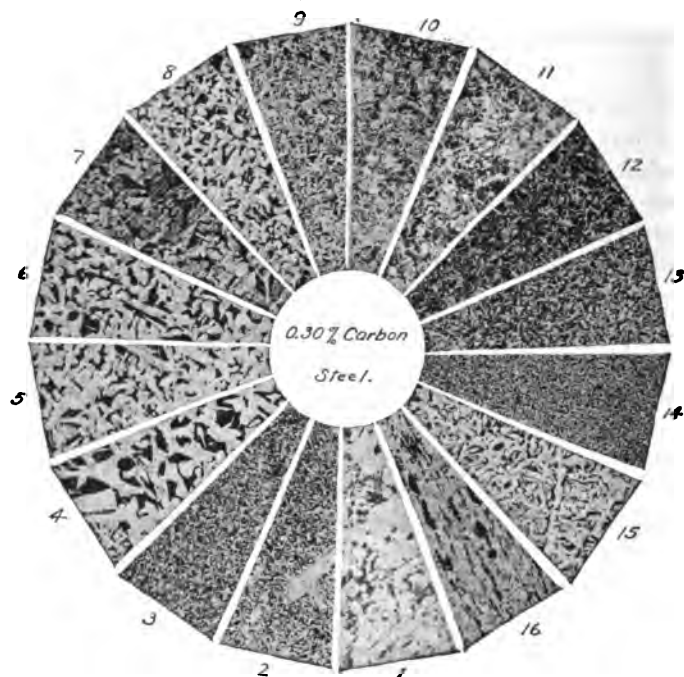
Bars 1 to 3 inclusive are pearlitic in structure. The white areas are ferrite (iron with certain impurities in solution) and the dark areas pearl-

* Assistant in Metallurgy, Harvard University.

† Non-member. Instructor in Physics and Electrical Engineering, Ohio University.

¹ Specimens were prepared by Charles Frohnert.

ite (the eutectoid of Fe_3C and ferrite). Bar 1 has a coarser structure than No. 2, and No. 2 in turn a coarser structure than No. 3. No. 6 is also pearlitic and is similar in every way to Nos. 1, 2, 3, except that its structure is very fine. No. 6 was the untreated specimen and represents the structure of the material as it came from the rolls. Nos. 4 and 5 have an entirely different structure. On passing through the critical range on cooling the ferrite has been rejected to the boundaries of the grains and the



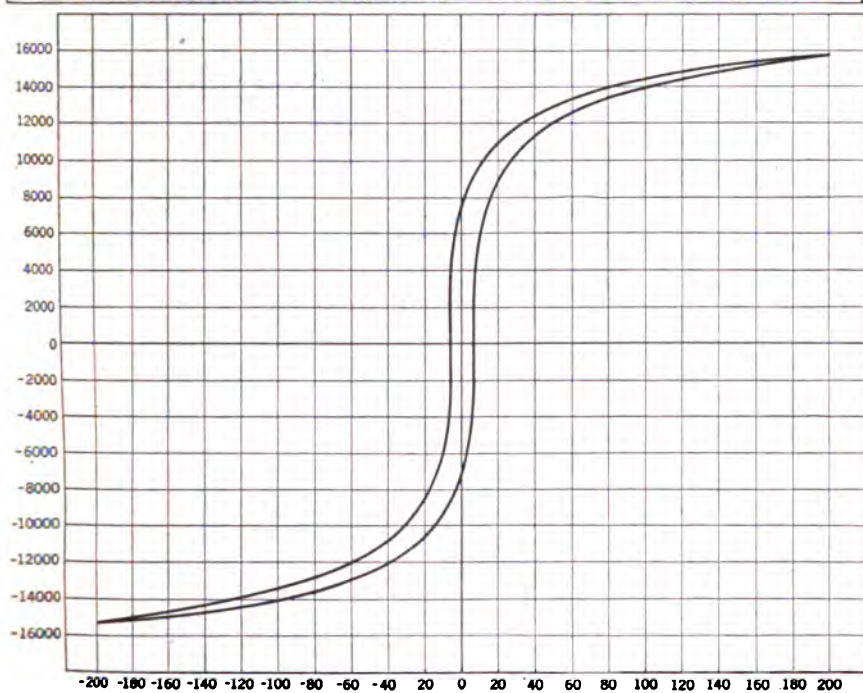
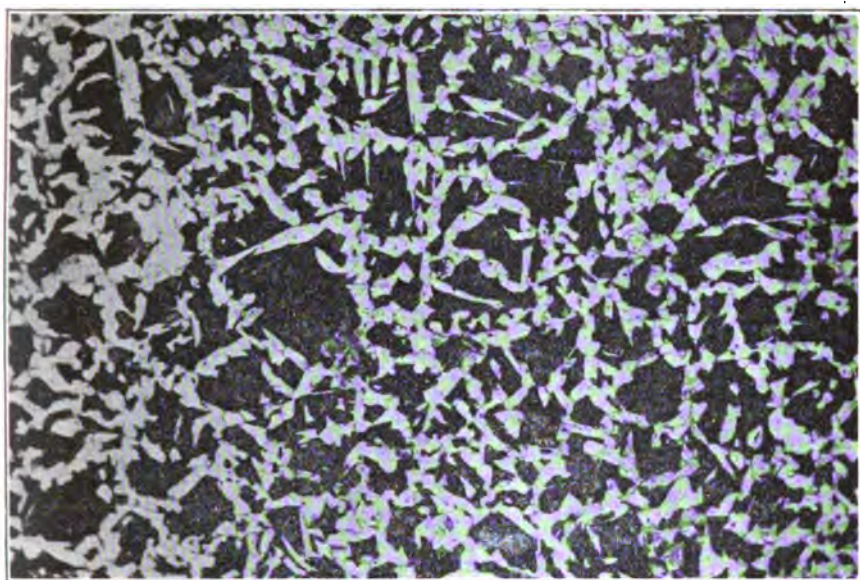
- | | |
|---|--|
| 1. Steel as cast. | 9-13. Heated above critical range, followed by cooling in air or oil, or heated above critical range, cooled in water or oil and reheated to 600° C. Ferrito-sorbite or ferrito-sorbite-troostitic structures. |
| 2. Steel cast and imperfectly annealed (remnants of ingotism). | 14. Forged and finished at low temperatures. |
| 3. Steel cast and properly annealed. | 15. Forged and finished at high temperatures. |
| 4-8. Heated to various temperatures above critical range for various lengths of time and slowly cooled in furnace. Ferrito-pearlitic structures of different degrees of coarseness. | 16. Cold worked. |

FIG. 1.—VARIOUS STRUCTURES OF MILD STEEL (0.30 PER CENT. C).

(A. SAUVEUR.)

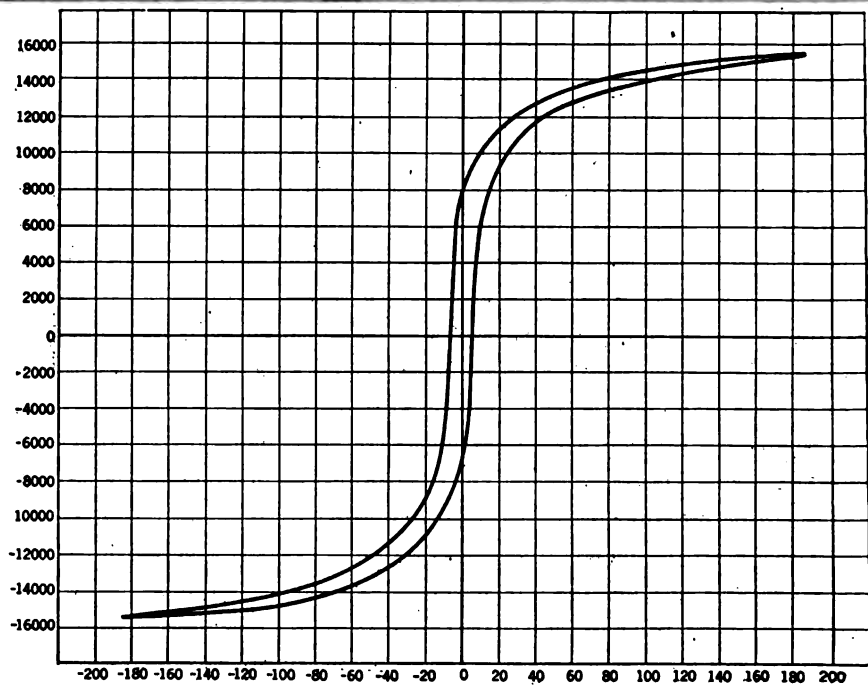
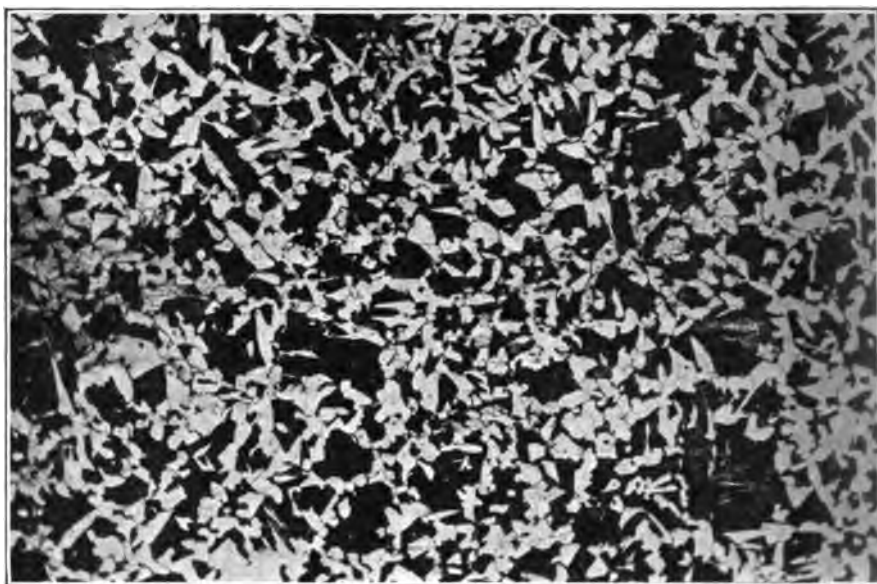
dark portions in these samples are a mixture of pearlite and sorbite. Sorbite can be regarded as poorly defined pearlite and is one of the transition constituents between austenite, existing above the critical range, and pearlite, below the range. No. 4 is more sorbitic than No. 5 and also has a larger grain.

Below each microphotograph (Figs. 2 to 7) is placed the B-H curve for the specimen together with some of the data which we regard as significant.



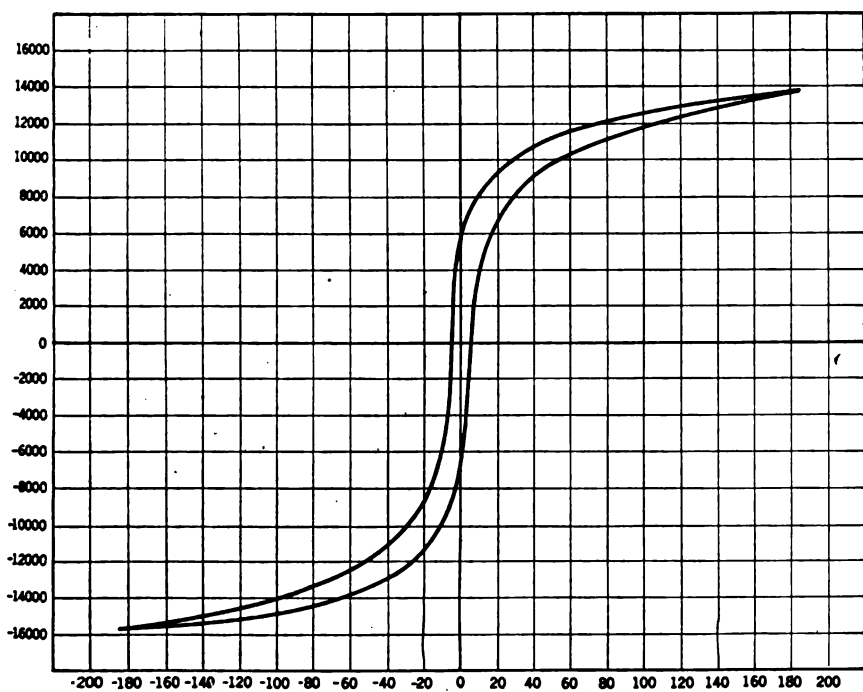
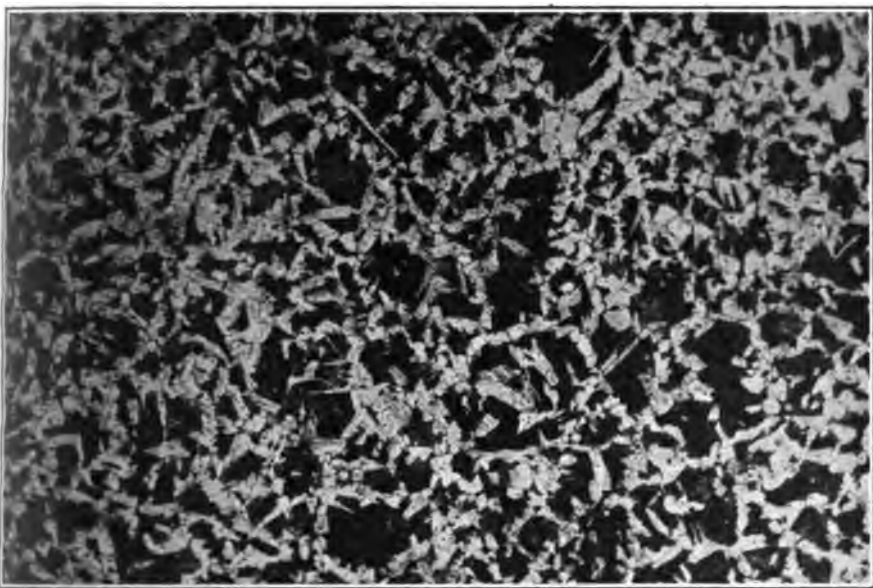
Hysteresis Loss, 17,280
 Residual B, 6,700
 Coercive Force, 3.65
 Hardness, 131

FIG. 2.—BAR 1.



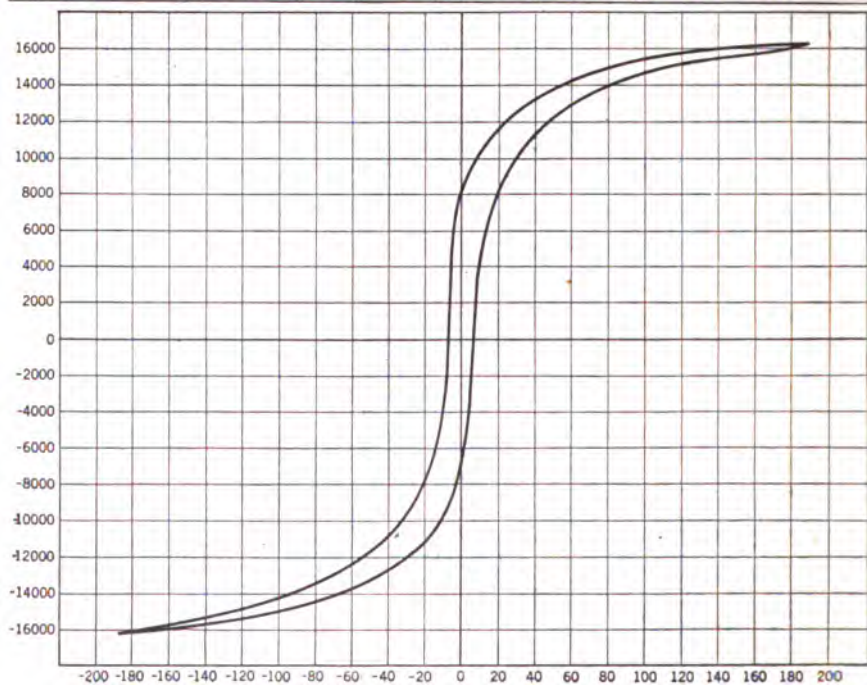
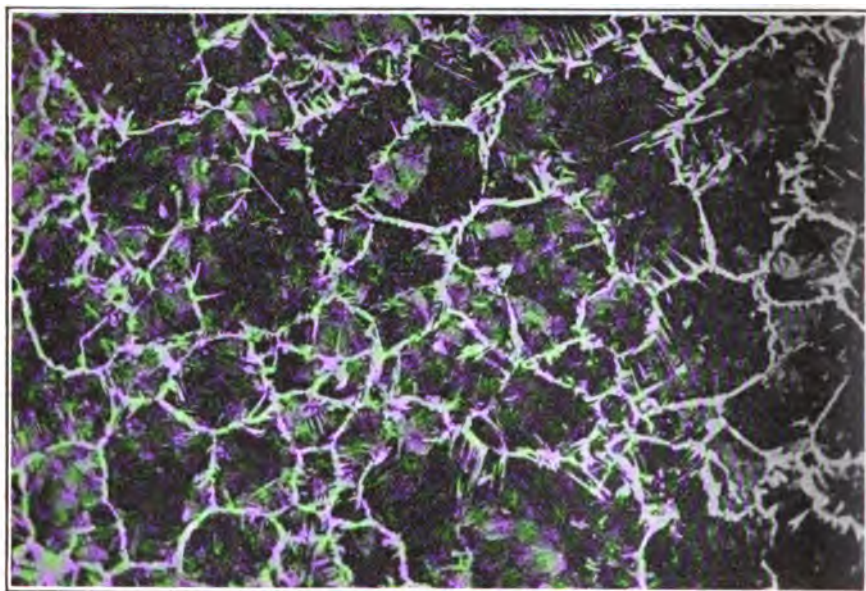
Hysteresis Loss, 18,240
 Residual B, 6,800
 Coercive Force, 3.70
 Hardness, 131

FIG. 3.—BAR 2.



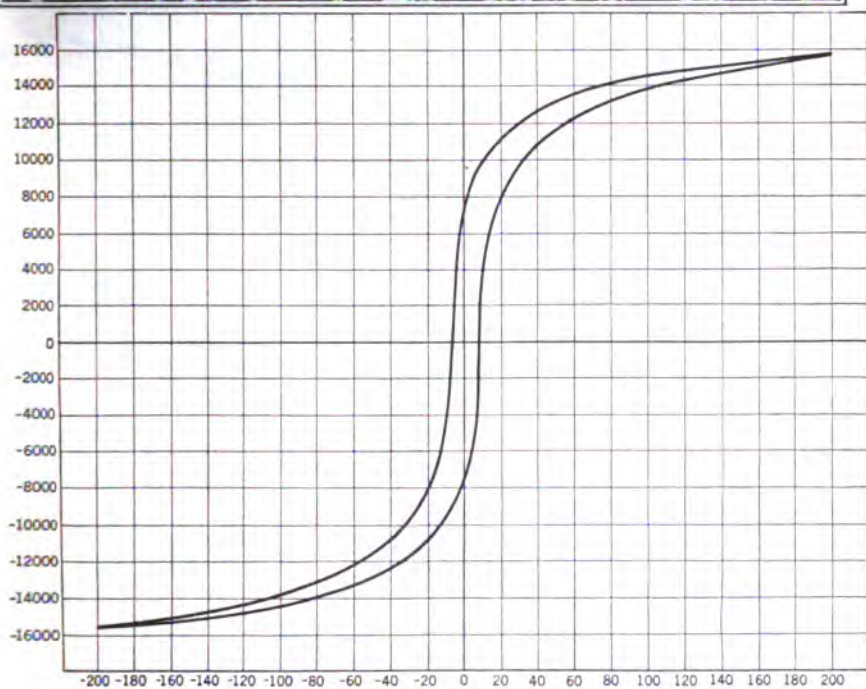
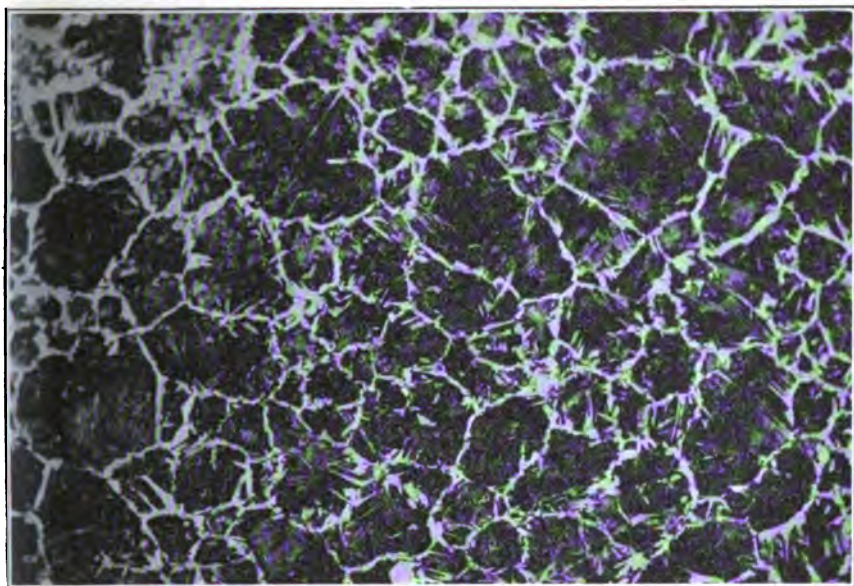
Hysteresis Loss, 21,920
 Residual B, 7,000
 Coercive Force, 3.72
 Hardness, 131

FIG. 4.—BAR 3.



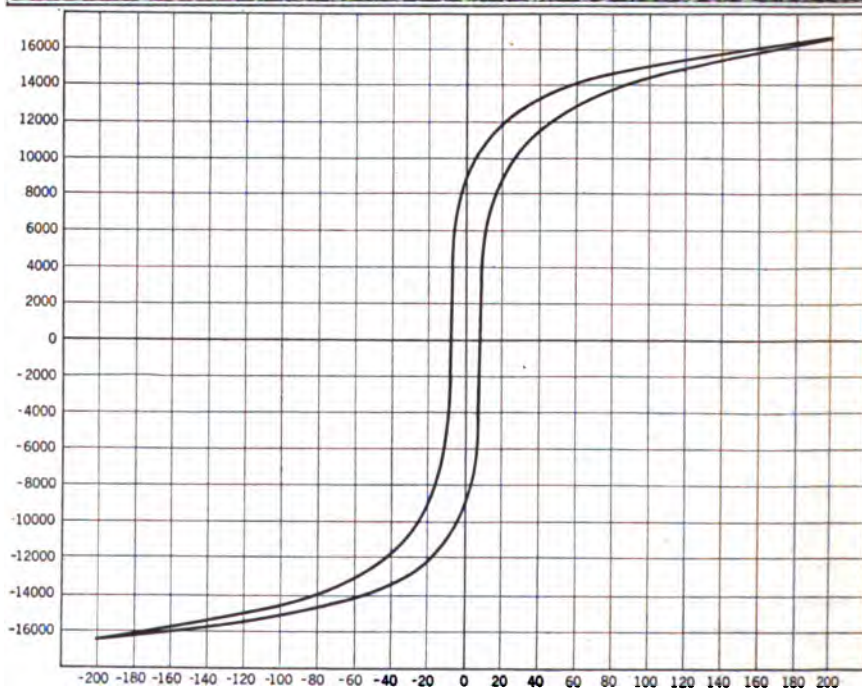
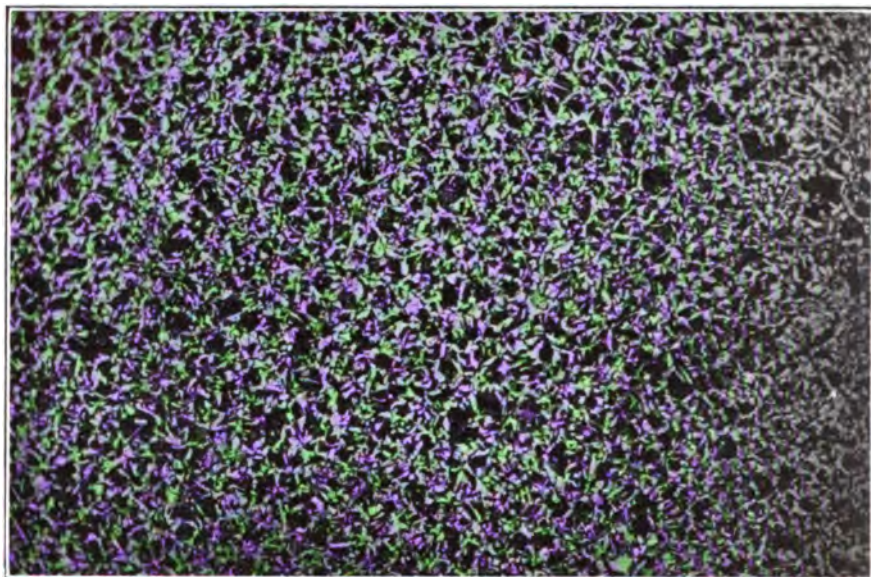
Hysteresis Loss, 26,240
 Residual B, 7.700
 Coercive Force, 7.00
 Hardness, 140

FIG. 5.—BAR 4.



Hysteresis Loss, 25,760
 Residual B, 7,300
 Coercive Force, 7.00
 Hardness, 131

FIG. 6.—BAR 5.



Hysteresis Loss, 29,120
 Residual B, 9,700
 Coercive Force, 7.50
 Hardness, 131

FIG. 7.—BAR 6.

Data

Bar No.	Loss in ergs per c.c.	Residual B	Coercive force	Brinell hardness
1	17,280	6,700	3.65	131
2	18,240	6,800	3.70	131
3	21,920	7,000	3.72	131
4	26,240	7,700	7.00	140
5	25,760	7,300	7.00	131
6	29,120	9,700	7.50	131

The hysteresis loss was determined as follows: The area of the loop inclosed by the B-H curve was measured by a planimeter. The coordinates being in absolute units, the resulting area as determined by the planimeter was multiplied by our scale constant, which was 40,000, and divided by 58, 58 c.c. being the volume of the bar tested.

Residual curves were not determined, but for each bar the residual induction B was determined after the magnetizing force H had reached its maximum. The coercive force was determined by measuring the magnetizing force H necessary to reduce residual B to zero.

Hardness of the specimen was measured by a Brinell machine and results are expressed in Brinell hardness factors.

Bars 1, 2, and 3 present a very interesting study. The only variable factors in these three bars are the relative size of grain and the coarseness of the striations of pearlite. Bar 1 has a large grain and the pearlite is coarsely laminated. Its hysteresis loss per cubic centimeter was 17,280. Bar 2 has a finer structure than No. 1, and shows a hysteresis loss of 18,240. Bar 3 is still finer than bar 2 and shows a loss of 21,920.

Bar 6, as before stated, is also pearlitic and differs from Nos. 1, 2, and 3 in having a much finer structure. The hysteresis loss in this bar rises to 29,120 ergs per cubic centimeter, which is 68 per cent. greater than the loss in bar 1. It is to be borne in mind that both are in the unhardened condition.

Bars 4 and 5 show an entirely different structure to any of the other bars. Bar 4 has a larger grain than No. 5 but the treatment employed to give the bar its coarse structure also produced a larger proportion of sorbite than is present in Bar 5. Bar 5 has a hysteresis loss 480 ergs less than bar 4, which seems contradictory to the results shown by the previous bars, as bar 5 has a finer structure than bar 4. The finer structure of bar 5, however, is only apparent and not real. The gross structure is finer—that is, the grains are finer—but the structure as revealed under higher magnification shows that bar 4 is composed of sorbite, laminated pearlite being almost entirely absent. In bar 5 considerable laminated pearlite is present.

Summary

1. Bars tested, six in number, carbon content 0.43 per cent. Nos. 1, 2, 3, and 6 were pearlitic and Nos. 4 and 5 were sorbitic.

2. With decreasing grain size Nos. 1, 2, 3, and 6 show a rise in their hysteresis loss, increasing residual B, and increasing coercive force.

3. Bar 4, comparable in grain size to bar 1, but differing from bar 1 in being sorbitic whereas No. 1 is pearlitic, shows an increase in its hysteresis loss of approximately 50 per cent.

4. In magnetic testing careful attention must be given to the internal structure of the metal undergoing test. A mere statement of its condition as hardened or unhardened, annealed or tempered, is idle and often misleading.

5. Our results lead us to infer that a minimum hysteresis loss could be obtained by the combination of large grains with coarsely laminated pearlite.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Plasticity of Clay and Its Relation to Mode of Origin

BY N. B. DAVIS,* ITHACA, N. Y.

(New York Meeting, February, 1915)

OUTLINE

- I. Introduction.
- II. Definition of Plasticity.
- III. Theories of Plasticity.
 - A. Structure of the clay particles.
 - (1) Fineness of grain.
 - (2) Plate structure.
 - (3) Interlocking particles.
 - (4) Sponge structure of particles.
 - B. Presence of hydrous aluminum silicates.
 - C. Molecular attraction between particles.
 - D. Presence of colloidal gelatinous matter.
 - (1) Review of theory of colloids.
 - (a) Suspension colloids.
 - (b) Emulsion colloids.
 - (2) Experiments.
- IV. The Formation of Clays.
 - A. Residual clays.
 - B. Transported clays.
- V. Summary.
- VI. Conclusion.

I. INTRODUCTION

WHILE working with a number of very sticky cracking clays from western Canada the writer became interested in a study of the cause of the excessive plasticity. This led to a review of the rather bulky literature on the subject and the finding that plasticity still remains to be satisfactorily explained. Most of these investigations seem to have been carried on solely from the chemical viewpoint, the geologic aspects having been ignored. In nearly every case, the investigators have worked with the higher-grade clays of low to medium plasticity, such as the kaolins, the excessively plastic ones having generally been neglected, partly because of their limited application, yet the author believes that these are

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the clays through which the problem should be attacked along with a study of the formation of clay deposits.

II. DEFINITION OF PLASTICITY

Numerous definitions of plasticity have been advanced from time to time by different writers and most important have been those proposed by Seger, in Europe, and Ries in America.

Seger¹ describes plasticity as "that property which enables a solid to receive a liquid into its pores, to retain the latter completely, thus enabling the mass, by pressure and kneading, to take any desired shape, to retain the shape unchanged on the removal of pressure and extraction of the liquid and the consequent change to the solid state. It is not found in all friable solids. Many cannot be molded when mixed with a liquid or on its removal will not possess sufficient cohesion of the particles to result in a strong body."

Other materials such as wax, tar, hot glass, etc., are described as plastic, but plasticity in this sense must not be confused with that in clays. Plasticity in the case of lead or hot glass is really malleability.

In explaining plasticity in the case of clay we have only to deal with that property developed on the addition of water and the subsequent drying to a body of some strength. Increased hardening and shrinkage usually accompany increase in plasticity.

III. THEORIES OF PLASTICITY

The various theories that have been advanced from time to time to explain the plastic properties of clay may be classified as follows:

A. Structure of the clay particles.

- (1) Fineness of grain.
- (2) Plate structure.
- (3) Interlocking particles.
- (4) Sponge structure of particles.

B. Presence of hydrous aluminum silicates.

C. Molecular attraction between particles.

D. Presence of colloidal gelatinous matter.

In the following pages the writer will attempt a review of the various theories which it seems to him have tended to lead up to the colloidal one.

A. Structure of the Clay Particles

In 1867 Johnson and Blake,² working on kaolinite, tried to explain plasticity as due to the fineness of grain, and plate structure of the kaolinite and other platy minerals present in clay.

¹ *Collected Writings*, vol. i, p. 533.

² *American Journal of Science and Arts*, 2d ser., vol. xliii, p. 357 (1867).

Clays in which they found bundles of kaolinite crystals, seemed to be of lower plasticity than those in which the bundles were broken up into their component cleavage plates. They give two analyses of kaolins of practically the same chemical composition, yet one is "fat" and the other is "short." From this they infer that the difference in the degree of plasticity is due to the difference in the state of division.

Since the publication of this view, similar explanations have been advanced by such writers as Biedermann and Hersfeld,³ and Cook⁴ in 1878, and more recently by Haworth and Wheeler,⁵ Atterberg,⁶ and Le Chatelier.⁷

Experimenting with fine grinding of minerals developing a plate structure, Wheeler reported that calcite and gypsum finely ground in a ball mill with water developed plasticity; furthermore, test pieces gave tensile strengths ranging from 100 to 350 lb. per square inch. Talc and pyrophyllite also showed some plasticity when treated in the same way, but the tensile strength was low. It is altogether probable that the high tensile strength reported in the case of calcite and gypsum was due to the fact that some of the material went into solution, and crystallized out as a cement on drying. This did not happen with the other two minerals, which are more resistant to solution.

Recently a clay was received at Cornell University by Dr. Ries which analyzed 98 per cent. dolomite, and under the microscope was seen to be made up of little rhombs about 0.008 mm. in diameter. It was as plastic as a kaolin and of low tensile strength.

Orton⁸ tried the effect of fine grinding on glass and noted a slight plasticity but no tensile strength.

R. T. Stull⁹ ground mica and found a "surprising increase in plasticity." He concluded that fineness of grain and plate structure were the cause of plasticity.

Grimsley and Grout¹⁰ ground samples of quartz, calcite, and mica, and noted a development of some plasticity in each case.

Precipitated barium sulphate mixed with 14 to 22 per cent. water was described as plastic,¹¹ but the material showed little tensile strength on drying. Minerals that split into scales or plates, such as barite, kaolinite, muscovite and other micaceous minerals, were described as

³ Bischof: *Die Feuerfesten Thone*, p. 23.

⁴ *Report on Clay Products*, New Jersey Geological Survey (1878).

⁵ *Clay Deposits*, Missouri Geological Survey, vol. xi (1896).

⁶ *Zeitschrift für angewandte Chemie*, vol. xxiv, No. 20 (May 19, 1911).

⁷ *van Bemmelen's Gedenkboek*.

⁸ *Brick*, vol. xiv, p. 216.

⁹ *Transactions of the American Ceramic Society*, vol. iv, p. 257 (1902).

¹⁰ *Clays, Limestones and Cements*, West Virginia Geological Survey, vol. iii, p. 46 (1905).

¹¹ Atterberg: *Loc. cit.*, p. 928.

strongly plastic if ground and elutriated so that the size of the particles did not exceed 0.002 mm. diameter. Materials like quartz, feldspar, and gypsum do not cleave so readily into plates and hence are not so plastic. Atterberg concluded that the plasticity of the European clays is due to the presence of mica meal in those of the north, and to kaolinite scales in those of the south.

Another German writer¹² believed that the particles were of extreme fineness and long or round in shape. The long thin particles gave greater contact surface thereby increasing surface tension and plasticity.

It is evident that the conceptions of what constitutes a plastic mass, held by the writers cited are at variance, except in the case of mica. Wheeler, and Grimsley and Grout, describe quartz, calcite, and gypsum as plastic, while Atterberg says they are not.

Rohland¹³ mentions the fact that clays ground in a drag mill are more plastic than those ground in a ball mill; because in the former the particles are brought to a smooth flat form, whereas in the latter the grinding produces angular or irregular forms. He adds that even if artificial means produced a structure similar to that developed in natural processes it would not develop plasticity in the clay but simply represent one of the factors favoring plasticity. Again, Leppla¹⁴ attributed plasticity to the small size of the crystal plates, to inflexibility without elasticity, and to the excellent cleavage and softness. Further, he was of the opinion that the adhesion between water and the crystal grains was stronger than the cohesion of the smallest cleavage pieces.

Two Russian investigators¹⁵ came to the conclusion that plasticity is due to the interlocking of the clay particles and depends for its best development on a proper mixture of very fine and coarse grains. At an earlier date, Olchewsky¹⁶ suggested the idea of interlocking hook-like particles and Termier,¹⁷ studying a series of French residual clays and shales, found numerous little hook-like aggregates to which he also attributed the binding power. A study of this by Wheeler in a number of Missouri clays showed, that in all the clays examined in which the hooks were found, the plasticity was low. By grinding to break up these hook-like crystals of kaolinite into their cleavage plates, plasticity was increased.

Daubrée¹⁸ found that long-continued grinding of feldspar developed some plasticity, and Olchewsky explained this as due to the porous spongy

¹² Linder: *Tonindustrie-Zeitung*, vol. xxxvi, No. 27, p. 382 (Mar. 2, 1912).

¹³ *Die Tone* (Wien, 1909).

¹⁴ *Baumaterialien Kunde*, vol. ix, p. 124 (1904).

¹⁵ Aleksiejew and Cremiatschenski: *Zap. imp. russk. techn. obschtsch.*, xxx (1896).

¹⁶ *Deutsche Topfei-und Zeigler-Zeitung*, No. 29 (1882).

¹⁷ *Comptes Rendus de l'Académie des Sciences*, vol. cviii, p. 1071 (1889).

¹⁸ A. S. Cushman: *Transactions of the American Ceramic Society*, vol. vi, p. 66 (1904).

structure of the finest particles caused by the removal of the alkalis in solution. Of all the earlier writers Olchewsky came closest to a true explanation of plasticity.

B. Presence of Hydrous Aluminum Silicates.

To fineness of grain and plate structure Vogt,¹⁹ Arons,²⁰ Bischof,²¹ Seger,²² and others have added the idea of the presence of hydrated aluminum silicates in clay as a cause of plasticity. It was noted that in general the temperature at which kaolinite lost its water of constitution was coincident with the disappearance of plasticity. This was on the assumption that kaolinite formed the basis of all clays—an assumption we now know to be incorrect.

Recent experimenters²³ have shown that all the combined water of the clay is not part of the clay base in the sense of water of hydration. Clay is a mixture of minerals, crystalloids and colloids, and hence has no definite dehydration temperature, water being lost more or less continuously. The clay substance of the impure clays was found to represent a type essentially different from that of the purer materials. Dehydration does not necessarily completely destroy the plasticity and hence "the combined water appears to have no direct connection with the phenomenon of plasticity." The conclusion that dehydration does not destroy the plasticity is questioned by Rohland,²⁴ who contends that it is lost at the temperature at which the combined water begins to be evolved, 590° to 620° C. It is probable that these authors have not the same conception of what constitutes a plastic mass or where plasticity ends. It is probable that some of the clays used by Brown and Montgomery contained considerable plate-like particles which made the material weakly plastic.

C. Molecular Attraction between Particles

Grout²⁵ was the first to develop this theory to any extent in connection with clays, although it was hinted at by Ladd,²⁶ Beyer,²⁷ and Leppla²⁸ before he took it up. He describes plasticity as due to the following factors:

1. The distance the clay particles can move on each other without

¹⁹ Ries: *Clays, Their Occurrence, Properties and Uses*, 2d ed., p. 47 (1908).

²⁰ Dammer: *Chemische Technologie der Neuzeit*, vol. i (1910).

²¹ *Die Feuerfesten Thone*.

²² *Collected Writings*, p. 69.

²³ Brown and Montgomery: *Dehydration of Clays, Technical Paper No. 21, U. S. Bureau of Standards* (Apr. 25, 1913).

²⁴ *Silicat Zeitung*, vol. ii, p. 30 (1914).

²⁵ *Clays, Limestones and Cements*, West Virginia Geological Survey, vol. iii, p. 54 (1905).

²⁶ *Bulletin No. 6a, Georgia Geological Survey* (1898).

²⁷ *Annual Report, Iowa Geological Survey*, p. 88 (1904).

²⁸ *Loc. cit.*, p. 124.

losing coherence, which varies with (a) shape and size of grain, (b) distance through the water film the particles will attract each other.

2. The amount of the coherence or resistance to movement, which varies with (a) the friction in the films, (b) the friction of the grains on one another.

He writes that fineness of grain, plate structure, and colloids are insufficient to explain plasticity and proposes a theory based on cohesion and adhesion, suggesting that the high plasticity of some clays is due to the greater attraction of the grains of those clays for water. Such an attraction depends on the constitution of the molecule and any change in the chemical constitution would afford a reasonable explanation of the fact that clays of similar chemical composition but of different molecular arrangement vary in plasticity.

Grout's molecular attraction comes down to nothing more than capillary attraction caused by the water in the interstitial spaces between the clay particles. It is well known that water wets different substances (minerals) to different degrees. We can add a small amount of water to a mass of dry ground spar, quartz, or calcite, and in each case the mass will hold together better than when dry simply because of the capillary force exerted by the water between the grains. When the mass is dried out it will fall down to a powder again simply because the capillary force no longer acts. If other material, such as a suitable colloid gel, is added another factor is introduced and the mass does not disintegrate. This brings the discussion up to the last theory.

D. The Presence of Colloidal Gelatinous Matter

Most writers at the present day believe that colloidal matter is the cause of plasticity, but our ideas of the colloidal state have been rather confused with the rapid advance of colloid chemistry and consequently the conceptions of colloids held by the various writers have differed somewhat, especially as regards clays.

In taking up this last theory it would perhaps be best to lead up to it through a discussion of the present-day conception of the colloidal state of matter.

(1) *Review of the Theory of Colloids*.—In the early '60's, Thomas Graham,²⁹ an English physicist, working on the phenomenon of diffusion of dissolved substances through organic membranes, found that certain substances passed through freely, while others did not do so, or at most, very slowly. In general he found that substances of pronounced crystal habit passed through membranes; while other substances, amorphous in character, such as gelatin, gum arabic, etc., apparently in solution, did not diffuse.

This led him to divide matter into two great classes, colloids and

²⁹ *Philosophical Transactions of the Royal Society*, vol. cli, pp. 183 to 224 (1861).

crystalloids. Further work along this line showed that substances, formerly considered soluble, could be obtained in what appeared to be true solution, yet they would not diffuse. These apparent solutions he called colloidal, or "sols."

He found that slight additions of electrolytes, either acids or bases, which did not react chemically with the substance in colloidal solution, tended to precipitate or coagulate it. These coagulated sols he called "gels" because of their jelly-like nature.

Graham worked with colloidal solutions of silicic acid, tungstic acid, chromium, aluminum, and ferric hydroxides. He was not the first investigator to prepare these sols, but he was the first to attempt a systematic investigation of their properties.

Since Graham's time great advances have been made in the study of the colloidal condition of matter, and colloid chemistry has developed as a branch of physical chemistry.

The term colloidal refers to a physical condition that may be taken on to a greater or lesser degree by all substances. Colloid chemistry is the study of the properties of substances in the finely divided state.

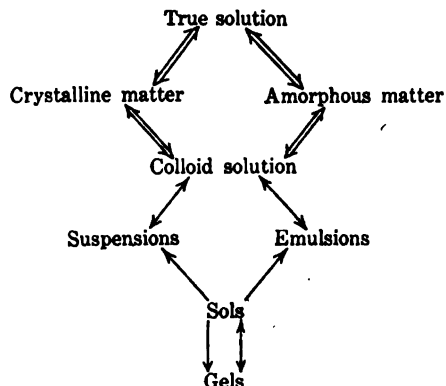
In considering the colloidal condition we have to think of two phases being present, namely the solvent and the colloiddally dissolved substance. Ostwald³⁰ in Europe, and E. E. Free³¹ in America, in papers published about the same time have classified the possible phases as follows, calling the solvent the dispersive phase and the colloiddally dissolved substance the disperse phase. Either phase may be solid, liquid, or gas.

Dispersive Medium	Disperse Phase		
	Gaseous	Liquid	Solid
Gaseous.....	Class 9: G + G. No real example as gases always are completely miscible.	Class 8: G + L. Fog <i>e.g.</i> , at the point of liquefaction of gases, atmospheric fog, etc.	Class 7: G + S. Smoke, cooled ammonium chloride, vapor, etc.
Liquid.....	Class 6: L + G. Foams.	Class 5: L + L. Emulsions.	Class 4: L + S. Suspensions; gels.
Solid.....	Class 3: S + G. Gas inclusions in minerals (pumice), or in metals.	Class 2: S + L. Liquid inclusions in minerals; occluded water, water of crystallization.	Class 1: S + S. Inclusions of solid particles in minerals; solid solution.
G = Gas L = Liquid S = Solid			

³⁰ *Kolloidchemische Beihefte*, vol. ii, pp. 94 to 97 (1910).

³¹ *Journal of the Franklin Institute*, vol. clxix, p. 424 (1910).

Most important are the liquid-solid and liquid-liquid systems, represented by suspensions and emulsions respectively. Any substance, amorphous or crystalline, may take on the colloidal condition, and there is a tendency toward an equilibrium between these three states and true solution. This may be represented diagrammatically as follows:



A large number of sols of the metals and their oxides, hydroxides, and sulphides are known and a study of them is becoming more and more important in explaining the formation of ore deposits. Many of them are present in clays and affect their properties to different degrees.

Certain substances, such as the gums, gelatin, agar, etc., are known only in the amorphous and colloid states, while other substances, such as the metallic oxides, etc., mentioned above, are known in all the different states. Sodium chloride is a substance we commonly know as crystalline and in true solution, yet under suitable conditions it can be prepared as a colloid.

There are a large number of organic compounds occurring in nature which can be dissolved as sols without any special precautions. As an evidence of their condition as sols, and not in true solution, it has been found that they have little or no effect on the boiling or freezing points of the solvents.

By the use of the ultra-microscope, invented in 1905 by R. Zsigmondy and H. Siedentopf, we have been able to observe the size of the particles in colloidal solutions. The particles of many sols have been measured and shown to be larger than the calculated size of the molecules. This, along with the inability to diffuse, and the non-effect on the boiling and freezing points, are the fundamental differences between colloidal and true solutions.

As Ostwald²¹ has stated, viscosity is another important property

²¹ *Transactions of the Faraday Society*, vol. ix, pp. 34 to 46 (1913). *Chemical Abstracts*, vol. viii, p. 848 (Mar. 10, 1914).

of fundamental importance in investigating the properties of the colloidal state, on account of the extreme sensitiveness of this constant toward very slight alterations in the conditions of colloid. The metal and sulphide sols have been found to have little effect on the viscosity of the solvent while organic sols have been found to show marked increase in viscosity. The organic sols are usually liquid (emulsions) and this has led us to consider that in colloidal solutions showing an increased viscosity the disperse phase is a liquid.

(a) Suspension Colloids (Suspensoids).—It was shown by Stokes in 1850, and since has been verified by others, that a small round particle falling in a liquid assumes a constant velocity; and as the radius of the particle becomes smaller, the rate of falling decreases. Hence, with very small particles, we may have a suspension that will appear stable for a long time. If the particles are irregular in shape, as they usually are, the velocity of fall is slower, due to increased resistance.

Besides gravity, other factors are to be considered in examining the stability. The most important, perhaps, is the electric charge taken on by the particles when suspended in the liquid medium. Clay particles suspended in water take on negative charges, and hence continually repel one another. This repulsion may be one cause of another phenomenon known as the Brownian movements, so named after their discoverer, Dr. Brown, an English botanist. The movement may be noticed in all colloidal suspensions. The particles are seen to be jumping around in all directions in the medium, yet not touching one another. This movement tends to keep the particles in suspension.

On the addition of many foreign substances, sometimes in very small amounts, changes in the suspension are noted. The organic colloids and strong alkalies tend to increase the stability and are known as protective colloids and deflocculators respectively. The neutral salts and acids cause flocculation and are known as flocculators. Much work has been done on the phenomenon of flocculation and deflocculation; and for a more detailed treatment of the subject the reader is referred to the work of E. E. Free,³² and to that of H. E. Ashley.³³

In 1874 Schloesing³⁴ applied the theory of colloid suspension to explain the action of clay suspensions on the addition of electrolytes. He allowed clay suspensions to settle for 27 days, and then evaporated the solution with the material remaining in suspension. The sediment from this liquid dried to a horny mass, and this he called colloidal.

Since Schloesing's time, P. Rohland in Europe, and H. E. Ashley in America, have been most active in experimenting along this line.

³² *Journal of the Franklin Institute*, vol. clxix, pp. 421-438 (1910).

³³ Technical Control of the Colloidal Matter of Clays, *Technical Paper No. 23* U. S. Bureau of Standards (November, 1911).

³⁴ *Comptes Rendus de l'Académie des Sciences*, vol. lxxix, pp. 376 and 473 (1874).

Ashley's work has been most important because of his attempt to control the colloid content of clays.

(b) Emulsion Colloids (Emulsoids).—It has been known for some time that the addition of small amounts of colloids belonging to the emulsion class greatly increases the stability of the suspension. Those that do this are given the special name of protective colloids. They are much less sensitive to electrolytes than the suspensions, and their effect is explained by assuming that each particle of the suspension is coated with a thin layer of the emulsion colloid, and then takes the electric charge of the latter. This action is important in explaining the plasticity of clay and will be referred to again.

The most important inorganic emulsoid is silicic acid. On the addition of hydrochloric acid to a sol of this acid it will set to a gel and the change is not reversible. It was thought at first that this gel represented a definite hydrate, but the work of Ramsay, Spring, van Bemmelen and others on dehydration has shown that it is not a definite hydrate. There is a continuous loss and gain of water as the humidity of the surrounding atmosphere varies. Similar work on the hydrates of iron, aluminum, chromium, etc., has shown that compounds of this nature are not definite hydrates, but in the same way take on and lose water with the varying conditions.

Löwenstein³⁵ experimenting with clays found that they lost water continuously. He found that as the temperature was raised gradually, there were no breaks in the dehydration curve to indicate definite hydrates of similar dehydration points in the clayey mixture. Reference has already been made to the work of Brown and Montgomery along this line.

Compounds of the silicic acid type have the important property of adsorption of other colloids and salts from solution; and because of this are known as adsorptive colloids. Such hydrous gel bodies are shown to be present in clays by their degree of absorption. Ashley's method of determining the colloid content of clay is based on this adsorptive phenomenon.

Rohland³⁶ claims to have been the first to demonstrate colloids as a cause of plasticity, but credit should be given Schloesing,³⁷ whose work remained hidden for so long.

Rohland has written much on the subject; he has developed partially the colloid theory of plasticity on the basis of the work of Graham and Schloesing.

In his first paper, published in 1902,³⁸ he pointed out that colloids,

³⁵ *Zeitschrift für anorganische Chemie*, vol. lxiii, No. 2, pp. 69 and 139 (July 23, 1909). *Chemical Abstracts*, vol. iv, p. 887 (Jan.-July, 1910).

³⁶ *Int. Mitt. für Bodenkunde*, vol. iii, p. 492 (1913).

³⁷ *Comptes Rendus de l'Académie des Sciences*, vol. lxxix, pp. 376 and 473 (1874).

³⁸ *Zeitschrift für anorganische Chemie*, vol. xxxi, No. 1, p. 158 (May 16, 1902).

as distinguished from crystalloids, are substances which possess shrinkage and cohesion on drying, properties associated with plasticity in clays. About the same time Ries³⁹ published his observations on plastic clays under the microscope and noted that the extremely minute spherical structureless particles might be of the nature of colloids.

In 1903, van der Bellen⁴⁰ accepted the theory of the presence of colloids, and states that the purest clays and kaolins are undoubtedly mixtures of aluminum salts of meta-silicic acid, the varying degree of plasticity depending on the presence of a colloid modification in the structure of the clay particles.

In 1904, Ries⁴¹ again writes that "it may seem doubtful whether mere interlocking alone is sufficient to account for the tenacity of the clay, and whether or not organic substances, included under the term of colloids and which are no doubt present in many clays, do not exert some cementing action."

The same year Cushman⁴² writes that clays are to be considered as aggregations of particles which fall under three heads: (1) non-plastic crystalline, (2) non-plastic amorphous, and (3) plastic colloid. He states that plasticity and binding power will not only vary with relative proportions of the above mentioned materials but also with the particular character and activity of the sort of colloid matter that happens to be present. Undoubtedly the principal substance of which all good clays are composed is the hydrated silicate of aluminum, or "kaolin,"⁴³ and yet the variation in binding power and plasticity is very great.

In his summary he presents the following points:

1. That both plasticity and binding power are merely manifestations of a colloid modification of matter which exists in rocks and clays.

2. That the activity of these useful qualities depends upon the characteristics of the special colloids that may be present, as well as upon their past history, and the modifying effect upon them of saline and organic solution.

3. That absorptive qualities of clays, such as their ability to stick when touched with the tongue, and as exhibited by their occasional use as "lakes," clarifying agents, etc., is to be ascribed to the same cause.

He adds further that the size and shape of the clay grains have yet to be considered in explaining plasticity.

The viscosity of the water films on clay grains⁴⁴ suggests that increased

³⁹ *The Clays and Clay Industry of New Jersey*, New Jersey Geological Survey, vol. vi, p. 83 (1904).

⁴⁰ *Chemiker-Zeitung*, vol. xxvii, No. 36, p. 433 (May 6, 1903).

⁴¹ *Transactions of the American Ceramic Society*, vol. vi, p. 82 (1904).

⁴² *Idem*, p. 72.

⁴³ Probably kaolinite is meant.

⁴⁴ Grimsley and Grout: *West Virginia Geological Survey*, vol. iii, p. 51 (1905).

viscosity may be due to the presence of "colloids." A plastic clay may be as one composed of finely divided plates or scales held together by mutual attraction and water film tension, aided by colloids if present.

In 1909, Rohland⁴⁵ summarized the theories of plasticity and sums up his own work as follows: "My investigations on the decomposition of clays in relation to their plasticity can briefly be summarized as follows:

"Those substances which form colloidal solutions in water develop more or less plasticity. Clay and porcelain bodies contain colloids of both organic and inorganic nature to a certain extent latent, the colloidal condition only being developed in water. The plasticity can be increased by both organic and inorganic colloids."

Soluble salts also seem to have an effect on the plasticity of a clay.⁴⁶ In general they are thought to increase plasticity. This has been discussed by Purdy and Moore,⁴⁷ who considered it an exceedingly probable assumption that it is the influence of the adsorbed salts that gives to a clay its plasticity. The theory of the cause of plasticity in clays which appeared most tenable to them was that of adsorbed, or otherwise held, salts, resulting in an enveloping liquid media of high surface tension.⁴⁸

Writing in 1912, Grout and Poppe⁴⁹ summarized their paper on plasticity as follows. "We find that the essential peculiarity of a plastic clay is the fact that the water used in wetting the clay is somehow rendered viscous. In explanation we have two well-known phenomena: molecular attraction and the action of colloids. Both are certainly present. Both are necessarily active. The calculation indicates that molecular attraction may be quantitatively sufficient, especially since the force is known to be variable and may be greater in clays than in any other mineral. Colloids alone would probably be quantitatively insufficient. No experiments have shown them capable of such effects in other mineral powders."

These authors have made the mistake of calling clay a mineral when in reality it is a mixture of minerals. The writer believes that colloid gelatinous films are quantitatively sufficient, when their relation to the solid clay particles in natural clays is properly understood.

Had Grout used the microscope more he would have noticed that clay particles are not clean-cut bodies, but, as will be shown later, are coated with gelatinous material, the inner edge of which is intergrown with the more rigid part (see Fig. 1). On adding gel material to a clean crystalline or amorphous powder or granular mass, the coatings have not

⁴⁵ *Die Tone* (1909).

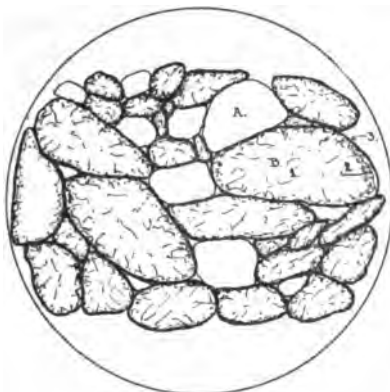
⁴⁶ Seger: *Collected Writings*.

⁴⁷ *Transactions of the American Ceramic Society*, vol. xix, p. 222 (1907).

⁴⁸ *Idem*, vol. xi, p. 582 (1909).

⁴⁹ *Idem*, vol. xiv, p. 80 (1912).

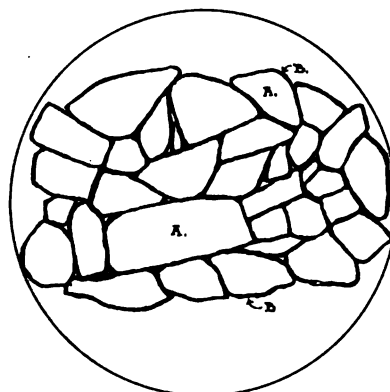
this intimate connection with the particles and hence such a mixture has not the same strength (see Fig. 2).



- A. Crystalline unaltered grains (quartz).
- B. Crystalline grains undergoing decomposition.
- 1. Interior of grains showing decomposition cracks.
- 2. Zone of more intense decomposition, with the formation of new, more stable, crystalline compounds and gelatinous modifications.
- 3. Gelatinous envelope.

FIG. 1.—NATURAL CLAY.

In recent papers Atterberg⁵⁰ and Rohland⁵¹ both consider colloid gels as a cause of plasticity in some degree. Atterberg lays stress on the



- A. Fresh undecomposed grains of a dry-ground powder.
- B. Coating of artificial colloid gel material on the grains (gelatinous silica, alumina, etc.).

FIG. 2.—ARTIFICIAL MIXTURE.

size and shape of grain, while Rohland holds that this has nothing to

⁵⁰ *Int. Mit. für Bodenkunde*, vol. iii, No. 4 (1913).

⁵¹ *Idem*, vol. iii, No. 6 (1913).

do with plasticity but that colloids are *the* cause. Without a doubt both are effective but the colloidal gels are most important.

If we are to understand that all very finely divided matter is in the colloidal state, then all clays contain colloidal material; hence the size of grain has something to do with plasticity, and the best plasticity is to be found in mixtures of colloidal and non-colloidal matter.

The chemical composition and physical properties of different substances in the colloidal gel state vary. For example, we may take a number of glues, which are colloidal substances, and note that the adhesive properties of each vary somewhat. In like manner the adhesive and cohesive properties of the various sticky colloidal substances in clays vary.

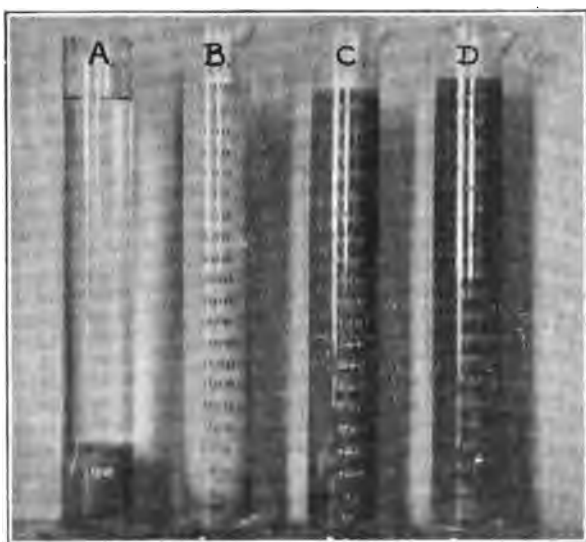


FIG. 3.—FOUR PLASTIC CLAYS AFTER 20 DAYS' SETTLING.

(2) *Experiments.*—As the writer stated in the introduction to this paper, he believes that the best way to study plasticity is to work with excessively plastic clays. Most of the work in the past has been done with the higher-grade clays, such as the kaolins, fire clays, ball clays, etc. Perhaps because of their few uses the excessively plastic cracking clays have been neglected.

A beginning was made on four excessively plastic clays from western Canada, 50-g. samples being taken and suspended in distilled water in 100-c.c. measuring tubes. After allowing the suspension to stand for 24 hr., samples were taken by means of a pipette at different depths, and the size of the largest particles in each case examined and measured under the microscope. One clay, on firing, showed considerable soluble

salts and the suspension settled in a few hours (see A, Fig. 3). The others remained in suspension for 20 days, when the particles, were measured again. This time the particles were so small in A and B that they could not be seen with a magnification of 600 diameters. However, the particles in C were visible, and measured 0.002 mm. in diameter. The sizes of the particles in the three clays remaining in suspension are shown in the curves of Fig. 4; and Fig. 3 shows the clays as they appeared in suspension at the end of 20 days.

A study of the curves shows that something is present in D causing larger particles to stay in suspension than in C and B. Also the particles remaining in suspension in C are larger than those in B. The pronounced effect of the "something" in D is shown by the size of the particles remaining in suspension after 20 days' settling.

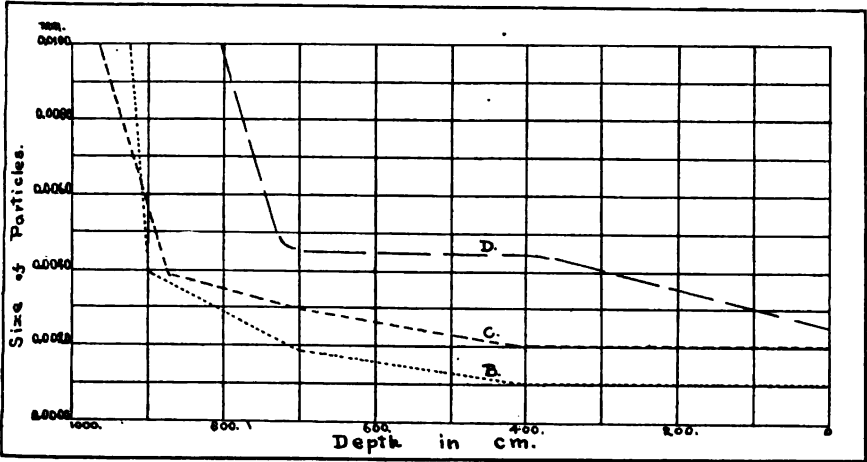


FIG. 4.—SETTLING CURVES OF SUSPENSIONS SHOWN IN FIG. 3.

This difference in the suspensions suggested a possible difference in the nature of the colloidal material present, so analyses for colloidal silica and organic matter were made. The relative depth of color in B, C, and D suggested that the percentage of colloidal organic matter increased toward D.

The following are the partial analyses:⁵²

	B	C	D
Colloidal SiO ₂	3.66	0.76	0.12
Carbonaceous matter (mostly colloidal)	0.21	0.26	1.10

In making the determination of colloidal silica in D the sodium carbonate extraction was dark in color, due to the colloidal organic matter,

⁵² Published by permission of the Director of the Geological Survey of Canada.

while in C it was much cleared, and in B comparatively clear. The effect of the large amount of colloidal organic matter in D was shown by the fact that, after 20 days, the particles in suspension could still be seen under the microscope. On the other hand it was most striking to find the colloidal silica percentage increasing toward B.

As stated in the discussion of colloids in general, the emulsoid colloids raise the viscosity of solvent. And as shown by Grout,⁵³ organic colloid gels have a much greater effect on plasticity than the inorganic colloid gels. This is well shown in the above analyses.

The organic colloid material may have the effect, as an emulsoid, of increasing the viscosity of the suspension, or may act as a protective colloid coating on the clay particles.

In these three clays plasticity is attributed to the colloidal silica in B, to the mixture of colloidal silica and organic colloids in C, and to the large proportion of organic colloids in D; without a doubt, other factors are working toward plasticity. Samples C and D both burn to a red, while B burns to a cream color, hence we may expect some colloidal iron oxide gel in C and D; and gelatinous alumina may be present in all three. However, organic matter and silica are most important.

According to Cameron and Bell,⁵⁴ little or no aluminum hydroxide forms in ordinary rock weathering. They examined several thousand soils from all parts of the United States and found aluminum hydroxide in only one sample, which came from California.

Recently an attempt was made to prove the presence of hydrous alumina in clays by recalculating analyses.⁵⁵ Such a method is of value in recalculating fresh rock analyses, but it cannot be applied to clays. Kaolinite was assumed as the basis for these calculations, it being considered the basis of all clays, an assumption we know to be incorrect.

The work of Bleining and Brown⁵⁶ on the viscosity of clay slips is of interest in this connection. They determined the relative viscosity of a number of clay slips, and found that the results roughly arranged the clays in the order of their plasticity. It would have been interesting to have had analyses for colloidal silica, organic colloidal matter, etc., on the clays used by them.

To test the effect of organic matter on a suspension, an attempt was made to duplicate natural conditions by putting into a ball mill a couple of pounds of Delaware washed kaolin, water, and an ounce of peat, with a couple of balls to stir. After mixing for 10 hr., the slip was drawn and

⁵³ *Clays, Limestones and Cements*, West Virginia Geological Survey, vol. iii, p. 80 (1905).

⁵⁴ *Bulletin No. 30, the U. S. Bureau of Soils*, p. 24 (1905).

⁵⁵ M. G. Edwards: The Occurrence of Aluminum Hydrates in Clays, Discussion by Ries, *Economic Geology*, vol. ix, No. 4 (June, 1914).

⁵⁶ *Transactions of the American Ceramic Society*, vol. xi, p. 604 (1909).

allowed to dry out. It was then mixed with water to a plastic mass. Comparing the resultant plasticity with that in a portion of untreated kaolin, it was found to have increased. Samples of the treated and untreated material were then suspended in water and allowed to settle. After 10 hr. it was observed that the untreated kaolin had settled clear, while the treated portion showed considerable matter in suspension. Test pieces showed that the shrinkage had increased from 4 to 8 per cent. There was no appreciable change in the tensile strength.

Another experiment was tried by mixing a ferric sulphate solution into powdered feldspar, and then precipitating gelatinous iron oxide around the grains by adding ammonia. The material showed some plasticity and the tensile strength was raised slightly.

Many similar experiments have been tried in the past by various investigators. Mention may be made of the work of Zimmer⁵⁷ in adding gelatinous silica to samples of kaolin. He found both plasticity and binding power increased. Grimsley and Grout⁵⁸ added 0.08 per cent. agar to two clays and found the plasticity increased. A similar experiment using gelatinous alumina required a larger amount (3 per cent.) of this gel to raise the plasticity as high as did the 0.08 per cent. agar. On drying in air, and wetting again, they found that the sample containing the alumina gel had dropped back to the original plasticity. From this they concluded that since plastic clays are not so injured by air drying, it is evident that such colloids do not explain plasticity. Mixtures of gelatinous alumina and silica acted in the same way. In these experiments the investigators failed in this way, because they were not duplicating natural conditions. Had they precipitated their gelatinous material in the mass and hence got it in intimate contact in thin films over the grains they would have found the plasticity to remain about constant.

As Ashley⁵⁹ has pointed out, it has been the custom for many years to add gelatinous matter to clay to improve its working qualities. Gelatin, glue, starch, gum arabic, dextrin, milk, and silicate of soda are gelatinous substances often added to pottery bodies. In India, cow dung, horse dung, rotten paper, gum, and starch are added to the local clays in making pottery.

Acheson's⁶⁰ patent to improve plasticity consisted of adding a colloid, tannin, to clay; and Auclair's⁶¹ patent consisted of dissolving out the colloid gels with alkali, and removing the sol by filter pressing, the

⁵⁷ *Transactions of the American Ceramic Society*, vol. iii, p. 36 (1901).

⁵⁸ *Loc. cit.*, p. 48.

⁵⁹ Technical Control of the Colloidal Matter of Clays, *Technical Paper No. 23*, U. S. Bureau of Standards, p. 82 (November, 1911).

⁶⁰ British Patent No. 7776, Apr. 3, 1907.

⁶¹ French Patent No. 372858, Dec. 22, 1906.

material remaining being pure crystalline ceramic paste. Keppler⁶² patented a process to improve plasticity by deflocculating with alkali, adding colloidal humus matter, and reprecipitating the whole with acid.

It is well known that boiling a clay with water for even a short time decreases the content of crystalline quartz and kaolinite, and increases the proportion of colloidal matter.⁶³

Hilgard⁶⁴ states that boiling serves to reduce the kaolinite ingredient of soils and clays to the amorphous, plastic, diffusible modification. By boiling a very slightly chalky pipe-clay for 85 hr. he succeeded in rendering it quite plastic. He was simply making conditions favorable for the mineral matter to go into solution.

In the experiments in which colloidal gel material was added to kaolin and feldspar, the plasticity was increased but the tensile strength was little affected, at least not to the extent expected when comparing the chemical composition of natural clay mixtures and artificial mixtures. This may be explained by a more careful study of Olchewsky's suggestion of a porous felty structure of the grains.

The examination of clay particles under the microscope shows that they are not crystalline all through. They show alteration to secondary products (crystalline, amorphous, and colloid), especially around the outside and following in on cleavage and other cracks.

All inorganic crystalline and amorphous substances are soluble to some extent in water, and this solubility is increased by the presence of salts or acids.

It is well known that the solubility in water of any substance is never zero, although it may be exceedingly small. Consequently kaolinite in contact with water dissolves to some slight extent and is immediately hydrolyzed. The solution thereby becomes unsaturated with respect to kaolinite; consequently more dissolves, which in turn is hydrolyzed, when still more can be dissolved, and so on. These two simultaneous processes may thus transform an appreciable amount of kaolinite, provided sufficient time be allowed—a condition which would usually obtain in nature.

This point of view is in harmony with conclusions reached by H. Gedroiz,⁶⁵ who asserted that aluminum hydroxide, ferric hydroxide, silicic acid, and kaolinite are, at the moment of their formation by weath-

⁶² *Chemical Abstracts*, vol. iii, p. 494 (Jan.-July, 1909). German Patent No. 201987, Aug. 4, 1906.

⁶³ Ashley: *Loc. cit.*, p. 84.

⁶⁴ *American Journal of Science and Arts*, 3d ser., vol. xvii, p. 211 (1879). Ref. to by Ashley, *loc. cit.*, p. 84.

⁶⁵ Ashley: *Technical Paper No. 23*, U. S. Bureau of Standards, p. 36 (November, 1911).

⁶⁶ *J. Exp. Landw.*, pp. 272 to 293 (1908).

ering, hydrosols, which stay as such or are coagulated, depending on the composition of the soil solution.

Mellor⁶⁷ found that the outlines of the solid particles of wet ground feldspar could be more readily stained with dyes than before the water treatment, indicating some change on the surface of the grains.

Of the common rock-forming minerals Buckman⁶⁸ gives the following list ranging from quartz, the least soluble, to nepheline, the most easily attacked:

- | | |
|----------------|---------------|
| 1. Quartz | 8. Talc |
| 2. Muscovite | 9. Hornblende |
| 3. Biotite | 10. Augite |
| 4. Orthoclase | 11. Apatite |
| 5. Plagioclase | 12. Olivine |
| 6. Epidote | 13. Leucite |
| 7. Serpentine | 14. Nepheline |

In the light of colloid chemistry we may consider minerals, or any mineral for that matter, in contact with water, tending to go into solution. Certain constituents will go to form new more stable crystalline compounds, and these, taking on the colloidal gel condition with ease, may develop in that state.

In a recent paper on the origin of laterite, A. Luz⁶⁹ has tabulated some of the possible changes as follows:

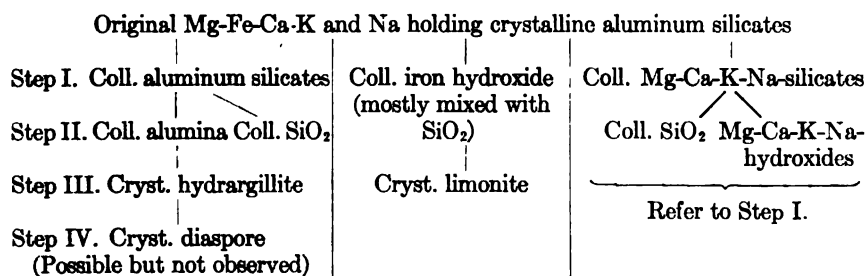


Fig. 1 represents a number of mineral grains very much enlarged, the interiors of which are criss-crossed by lines of alteration while the outer rims show, first, a zone of solution; second, a zone of colloid gels and new crystalline material; and third, a water film. In this diagram the writer sees an explanation of the impossibility of duplicating the degree of plasticity and binding power of clays by making up mixtures of fresh crystalline and colloid gel matter in the laboratory. There is not the same intimate connection between the gelatinous matter and the crystal-

⁶⁷ *Transactions of the English Ceramic Society*, vol. v, p. 72.

⁶⁸ *Transactions of the American Ceramic Society*, vol. xiii, p. 346 (1911).

⁶⁹ *Kolloid Zeit.*, p. 86 (1914)

line skeleton as in colloid and crystalline mixtures in nature. We have not been able to duplicate the time element as yet.

Ashley⁷⁰ describes an experiment in which he shook up portions of Georgia sagger clay in 100-c.c. graduated tubes, with water to make 100 c.c. After shaking for an hour, they were allowed to stand for some time and then most of the clear liquid siphoned off to be replaced by fresh water. This was repeated about 80 times and it was found that the bulk of the sediment had increased. He found this in agreement with the observations that the sediment becomes bulkier on prolonged contact with water, and indicates that the amount of colloidal material increases under these conditions. This is exactly what happens in nature when a residual kaolin of low plasticity is worked over by water, transported, and deposited as a sediment. This will be taken up in more detail under the formation of clays.

IV. THE FORMATION OF CLAYS

In taking up the study of the formation of clays as an aid in explaining plasticity it would perhaps be best to follow some genetic classification. The author recognizes how hard it is to get a complete classification to cover all possible cases. As this is intended more as a general study of the question it would perhaps be best to use as simple a classification as possible. That suggested by Ries⁷¹ comes nearest to this and with minor changes will be used, as follows.:

- A. Residual clays.
 - 1. Kaolins or china clays.
 - (a) Veins derived from pegmatite.
 - (b) Blanket deposits.
 - 2. Buff- and red-burning residuals, derived from different kinds of rock, igneous, metamorphic, or sedimentary.
- B. Transported clays.
 - 1. Deposited in water.
 - (a) In running water.
 - (1) Flood-plain clays, usually impure and sandy.
 - (2) Glacial stream clays in drift and always stony (boulder clays).
 - (b) In still water.
 - (1) Lacustrine or lake clays.
 - (2) Estuarine clays.
 - (3) Marine clays.
 - 2. Wind-formed deposits, loess.

⁷⁰ *Technical Paper No. 23, U. S. Bureau of Standards*, p. 48 (November, 1911).

⁷¹ *Clays, Their Occurrence, Properties and Uses*, 2d ed., pp. 27, 28 (1908).

A. *Residual Clays*

Residual clays may be defined as those clays resulting from the weathering of rock in place. Rock may be igneous, sedimentary, or metamorphic.

In countries that have suffered glaciation, clays of this type are rarely found, but in those parts of the world that have escaped glacial action, or where erosion is not too fast, residual clays are common. In general, then, most residual clays are found in warm countries.

The processes of weathering, working toward the formation of this type of clay, may be classified as follows:

1. Mechanical or disintegration.
 - (a) Changes in temperature.
 - (b) Work of plants and animals.
2. Chemical or decomposition.
 - (a) Solution.
 - (b) Hydration.
 - (c) Oxidation.
 - (d) Deoxidation.
 - (e) Recombination.

As a result of these processes acting on rocks of all kinds, residual clays are formed which may be classified under two sub-heads, depending on their uses or burning qualities. Those resulting from the disintegration and decomposition of granitic and gneissic rocks high in the feldspathic minerals, and low in the iron-magnesium minerals, are the high-grade kaolins or china clays. Those resulting by the same process acting on more basic igneous and metamorphic, as well as on the impure sedimentary rocks, are the lower-grade clays, ranging from those used for making fire brick to the poorest common-brick clays.

Kaolins.—The theories advanced to explain the formation of kaolin from feldspathic rocks may be classified as follows:⁷² (a) simple weathering, (b) post-volcanic emanations, (c) ascending spring waters, containing CO₂, (d) waters draining from swamps or peat bogs, (e) sulphuric acid solutions, (f) by the alteration of sericite.

There are examples of kaolin deposits formed by all the processes, but the most common is that by simple weathering.

Kaolins formed by all the above methods are of low plasticity, and it is lowest in those in which water played a minor rôle, as in post-volcanic emanations.⁷³

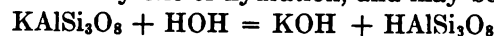
The chemical reactions that go on are not definitely known in any case, and are extremely complicated. Little more than a shrewd

⁷² Ries: *Transactions of the American Ceramic Society*, vol. xiii, p. 51 (1911).

⁷³ Jackson and Richardson: *Transactions of the English Ceramic Society*, vol ii, p. 59.

guess can be made, based on a knowledge of the original minerals and some of the residual products.

By whatever process a kaolin deposit may have been formed, it is generally conceded that feldspar was the principal original mineral. The alteration in feldspar has been shown by Cameron⁷⁴ and others to be essentially one of hydration, and may be represented as follows:



(orthoclase)



(pyrophyllite)



(kaolinite)



(diaspore)(?)

The silica may be taken away largely in the sol state, or may remain as a gel, due to coagulating agents being present. It may crystallize out, in part, and appear as quartz.

The alkali may be taken away in solution, or may enter into the formation of muscovite.

Because of the small amount of colloidal gel matter formed, the plasticity is low. The mineral grains are more crystalline than in any other type of clay.

Due to infiltration, organic colloid matter may be present in small amount. It is never very much, except perhaps near the surface of the deposit, where, as Ries⁷⁵ has noted, greater plasticity is developed. This is not to say that organic colloidal matter is the sole cause of the increase in plasticity at the surface, but it is one cause.

Kaolins do not vary much in their chemical composition and physical properties, as shown by the table on the following page, compiled from the report by Watts.⁷⁶ Yet some of them do seem to have present a substance which increases the tensile strength and the airshrinkage at the same time, properties increased in effect by the presence of gels.

The plasticity of residual kaolins is low, and may be attributed to a mixture of fine and coarse plates or particles, and minute amounts of gelatinous silica, alumina, organic colloids, etc.

Buff- and Red-Burning Residuals.—By far the most commonly occurring residuals are the buff- and red-burning clays, derived from all kinds of rocks, except those from which kaolin develops. Passing in igneous rocks, from granites through gabbros to the most basic periodotites, the mineral composition changes from feldspar-mica-quartz, through

⁷⁴ *Bulletin No. 30, U. S. Bureau of Soils.*

⁷⁵ *Maryland Geological Survey*, vol. iv, p. 463 (1902)

⁷⁶ Mining and Treatment of Feldspar and Kaolin in the Southern Appalachian Region, *Bulletin No. 53, U. S. Bureau of Mines*, p. 89 (1913).

Analyses and Tests on Washed Kaolins^a

	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	TiO ₂	CaO	MgO	BaO	Na ₂ O	K ₂ O	H ₂ O	Total Shr. at 110° C.	Cone II Fire Shr.	Ten. Str.	Ref. Value	Total Shr.
I. Kindland.....	50.64	35.57	0.25	0.03	Tr.	Tr.	0.07	0.08	1.70	11.90	100.24	4.40	8.0	1670°	14.2
II. Buchanan....	46.30	39.06	0.20	0.04	Tr.	Tr.	Tr.	0.11	0.60	13.77	100.08	5.40	28.5	1730+	19.3
III. Forest Hill..	49.20	37.58	0.17	Tr.	Tr.	Tr.	Tr.	0.13	0.47	12.53	100.08	4.00	16.0	1730	13.7
IV. Dillsboro....	46.95	37.73	0.15	0.05	Tr.	Tr.	Tr.	0.18	0.60	13.99	99.65	6.10	24.0	1730	18.8
V. Piedmont.....	48.50	37.35	0.85	Tr.	Tr.	Tr.	Tr.	0.32	1.02	12.00	100.04	4.40	16.5	1710	12.5
VI. Gurney.....	44.00	40.79	0.11	Tr.	Tr.	Tr.	Tr.	0.07	0.55	14.72	100.24	5.40	27.5	1730+	17.3
VII. McGuire....	46.35	39.00	0.30	Tr.	Tr.	Tr.	Tr.		0.50	14.00	100.15	5.70	10.50	1730+	18.2
VIII. Baby.....	46.90	38.60	0.25	Tr.	Tr.	Tr.	Tr.	0.26	0.39	13.80	100.20	6.25	21.5	1730+	19.75
IX. Smith.....	48.50	37.69	0.31	Tr.	Tr.	Tr.	Tr.	0.02	0.91	12.55	99.53	6.80	20.5	1670	18.8
X. Southern.....	46.67	39.07	0.11	0.02	Tr.	Tr.	Tr.	0.11	0.25	13.22	99.45				
XI. West.....	48.92	36.37	0.37	0.02	Tr.	Tr.	Tr.	0.11	0.29	12.70	98.78	7.00	24.0	1730	25.0
XII. Sprucepine..	45.20	38.45	0.45	Tr.	Tr.	Tr.	Tr.		0.65	14.80	99.55	4.00	8.0	1730	16.6
XIII. Tolley.....	46.35	38.80	0.25	Tr.	Tr.	Tr.	0.03		0.41	14.00	99.84	5.40	8.0	1730	16.3
XIV. Bryson.....	46.95	37.24	0.40	0.05	Tr.	Tr.	0.03	0.24	0.49	14.10	99.47	4.00	14.0	1730	16.8

^a From Bulletin No. 53, U. S. Bureau of Mines.

feldspar-amphibole-pyroxene to amphibole-pyroxene-olivine, with other accessory minerals such as the micas present in lesser amounts. This means a loss of alkali-aluminum silicates, and an increase in iron-magnesium silicates, which are known to be less resistant.⁷⁷

The same may be said for metamorphic rocks, except the feldspar gneisses, which yield kaolins. The more basic hornblende gneisses, slates, schists, etc., all come under this general statement.

The sedimentary rocks vary much in mineral composition and often the mineral particles are so small as to make it impossible to determine their character. Resulting as they have, mainly from water transportation, the colloidal condition is already developed to some degree, and its extent will depend on the age and past history of the deposit.

The limestones yield products ranging from fire clays to the poorest common-brick clays. Sandstones seldom yield clays, while shales usually do, since they are simply hardened clays. Here we are on the border between residual and transported clays, and shales may be included under both. In brick making, shales are often simply ground up and by this artificial means reduced to a soft clay again. However, it is well known that weathering gives a much more plastic material, due to the rejuvenation of the colloid gels and formation of fresh gels.

That mineral composition of the parent rock has much to do with the resultant plasticity in the clay may be shown by the following quotation:⁷⁸ "Clays obtained from gabbros and similar basic or dark-colored rocks are usually highly plastic, often deeply ferruginous and in many cases fine grained." In the same report a ferruginous clay of this type is described from near Catonsville, Md. It is a clay of great plasticity and wonderful tensile strength. Granites in the same general area yield plastic ferruginous clays, but they are less plastic, and less ferruginous usually, than those derived from the more basic rocks.

Residual clays in the State of Missouri are largely derived from limestones and shales.⁷⁹ The limestone-derived clays are yellow to brown or red in color, are excessively plastic, and consequently can seldom be used because of cracking in drying and burning.

Many other examples of plastic clays of the residual type might be given to show that it is in those clays high in the constituents iron oxides, silica, alumina, etc., commonly assuming the colloidal gel form, that plasticity is best developed.

B. Transported Clays

By far the greatest number of clays used in the ceramic industries are the transported clays. They are more widely distributed than the

⁷⁷ Buckman: *Transactions of the American Ceramic Society*, vol. xiii, p. 347 (1911).

⁷⁸ Ries: *Maryland Geological Survey*, vol. iv, p. 463 (1902).

⁷⁹ H. A. Wheeler: *Clay Deposits, Missouri Geological Survey*, vol. xi (1896).

residual type and vary in purity from sedimentary kaolins through ball clays, stoneware clays, fire clays to common-brick clays and the generally useless gumbos. They represent mixtures of the decomposition products of rock of all kinds carried by water in streams and rivers and deposited on flood plains, in glacial drift, in lakes and swamps, in estuaries, and under marine conditions.

The materials making up these clays not only suffer the action of water and other solvent agents, but also the attrition of particle on particle while being worked over by the stream.

It is interesting here to quote from the observations of George H. Cook⁸⁰ published in 1878. He was reporting on the clays of the State of New Jersey and had in mind the transported type so common in that State. "Some clays appear to consist of well-defined crystalline forms; others show a few of these in a mass of fragmentary shapes; others still seem to be wholly made up of irregular forms and exceedingly fine particles of matter. A satisfactory explanation of these different conditions is that the more finely divided clays are those which have had their crystalline forms broken up, either wholly or in part by the several agents that have moved them from the place of their origin to their present location, while those in which these forms still abound have not suffered the same constitutional derangement. Now, it has been observed that the former class of clays are more plastic than the latter. And a further observation is that by breaking up these crystalline forms and rendering them finer, the plasticity was promoted."

A very important fact connected with the origin of clay, and one which has great influence upon the plasticity, is that the farther clayey material is transported, during which it is constantly exposed to the abrading and comminuting action of the particles upon themselves as well as upon the banks and bottoms of streams over which they are carried, the finer it is ground and the smaller the scales into which kaolinite particles are broken.⁸¹ The smaller the crystalline plates of kaolinite are divided the more plastic is the clay, so those clays that have been subjected to severe wearing action are likely to be extremely plastic, whereas those that have had little or no wear, as those formed *in situ*, are but slightly plastic. This is well shown in the glacial and loess clays, where mechanical disintegration and abrasion of the clay particles have been great. Such clays are very plastic, though possessing a very moderate amount of kaolinite; while many of the china and flint clays which have not been transported possess almost no plasticity.

(a) Clays Deposited in Running Water.—Upstream on flood-plain flats are formed deposits of material carried by the stream from its

⁸⁰ *Clay Deposits of New Jersey*, Geological Survey of New Jersey, p. 287 (1878).

⁸¹ Wheeler: *Clay Deposits*, Missouri Geological Survey, vol. xi, p. 24 (1896).¹

source. This material may be made up of the wash of all kinds of rocks or only from certain kinds, depending on the geology of the drainage basin. In areas where transportation is fast and the original rock is granitic, the most of the material will be sandy, granular in character, and on the flood plains the deposits will be mostly of a sandy nature with some clayey material. Such a deposit would show little or no plasticity. The feldspathic minerals and quartz are least readily attacked by solution and hence would remain nearly intact. However, where the more basic iron-magnesium minerals are present, the resultant flood-plain deposit shows more plasticity. In either case had the original rock suffered residual weathering before transportation the chances for the formation of colloidal gel products would have been increased.

In the State of Florida a clay occurs that is probably of this type. It is the well-known Florida ball clay. It is supposed to have resulted from the transportation and deposition of material from a feldspar-granite area. It consists of a mixture of white clay and well-rounded quartz pebbles, the latter forming 65 to 75 per cent. of the entire mass.⁸² From this natural clay, the fine material is washed out and put on the market as Florida ball clay or kaolin.

In chemical composition the washed product is almost exactly that of the washed kaolins from residual deposits. The ball clay is highly plastic while the residual kaolins are of low plasticity. The only other difference between them is the method of their formation; the attrition and extra water action the ball clay was subjected to in transportation. The explanation of the difference in plasticity, then, must lie in the action of water on the grains.

The character of the clay depends largely upon the nature of the rock formations over which the stream and its tributaries flow.⁸³ As a concrete example, the alluvial clays along the large streams in the Piedmont Plateau contain a large percentage of quartz and mica sand and other undecomposed minerals derived from igneous rocks, and are simply residual kaolin made plastic by an increase in gelatinous matter in the processes of transportation and deposition.

As a rule flood-plain deposits are of lower grade and are usually used for common brick. Such deposits are made up of material from more basic rocks than are described above. A good example is that occurring near Edmonton, Alberta.⁸⁴ It consists of alternating layers of sandy, silty clay, and occasional pockets of gravel. It is used for common soft-mud and dry-press face brick. The run of bank is plastic, gritty, and calcareous. It burns to a light red at cone 03. The material was de-

⁸² Ries: *Clays, Their Occurrence, Properties and Uses*, 2d ed., p. 334 (1908).

⁸³ Otto Veatch: *Bulletin No. 13, Georgia Geological Survey*, p. 29 (1909).

⁸⁴ Ries: *Clay and Shale Deposits of the Western Provinces, Memoir No. 24-E, Geological Survey of Canada*, p. 37 (1912).

rived from the areas of sedimentary rocks drained by the Saskatchewan River.

The flood-plain deposits along the largest rivers in the northern part of Missouri are very plastic, sticky, black clays and are known as "gumbos."⁸⁵ Physically they are very fine grained, extremely plastic and tenacious, cracking badly in drying. They show very high tensile strengths, ranging from 270 to 410 lb. per square inch. They have a low specific gravity, 1.98 to 2.05, and contain from 2 to 5 per cent. organic matter. In these clays the organic matter in gelatinous form is the cause of the excessive plasticity.

Yet another example may be taken from Ries's⁸⁶ report on Texas clays. This gives analyses of two clays from Harrisburg, Tex. They form different beds in the same bank, and although their chemical composition is practically the same their physical properties vary widely.

	I	II
SiO ₂	80.39	80.84
Al ₂ O ₃	9.82	8.09
Fe ₂ O ₃	2.88	2.25
CaO.....	0.42	1.44
MgO.....	0.45	0.26
Na ₂ O.....	0.19	0.10
K ₂ O.....	Tr.	Tr.
TiO ₂	0.35	0.78
H ₂ O.....	3.11	6.00
	97.61	99.76
Water required for mixing, per cent. . .	18.7	19.8
Average tensile strength, lb. per sq. in..	188	275
Air shrinkage.....	4.8	8.6
Plasticity.....	fair	high
Drying.....	no cracks	cracks
Absorption cone 5.....	15.64	8.19
Steelhard.....	cone 9	cone 5

"A more interesting contrast could hardly be desired, and it forms no exceptions." Can we attribute the high plasticity, high shrinkage and the tensile strength in example II to plate structure, molecular attraction, or mere fineness of grain? Most certainly not. The presence of colloidal gel coatings on the mineral grains is the only logical explanation. Another point is significant. As Zimmer⁸⁷ has pointed out, pottery mixtures containing silica in the gelatinous form fuse at lower temperature than mixtures containing crystalline silica or quartz. A similar effect is shown in the burning of these two clays. Clay II becomes steel-

⁸⁵ Wheeler: *Clay Deposits*, Missouri Geological Survey, vol. xi, p. 542 (1896).

⁸⁶ *Clays, Their Occurrence, Properties and Uses*, 2d ed., p. 64 (1908). *Bulletin* No. 102, University of Texas, p. 224 (1908).

⁸⁷ *Transactions of the American Ceramic Society*, vol. iii, p. 25 (1901).

hard at a lower temperature than I. Also the absorption at cone 5 is lowest in the case of clay II. Hence it is fair to argue that the difference in burning qualities is due to the presence of more colloidal gel matter in II than I. Organic matter should have been determined and also colloidal silica.

Glacial clays are not often used because of their limited extent and stony character. They are often called boulder clays. The fine material has often been described as "ground rock flour," but that phrase is misleading. In its formation it was ground wet and subjected to water action for some time, a proper condition for the formation of the colloidal state.

As in other transported clays, the material has been gathered from various kinds of rock. The clay formed by material from basic rocks is more plastic than that from granite areas.

(b) *Clays Deposited in Still Water.*—Clay deposits formed in lakes and swamps form an intermediate group between the river and deep-water clays. They are often of limited extent and basin shaped, and again may be quite extensive. Frequently they consist of alternating clay and sand beds. They are common in glaciated regions and are often of glacial origin.

Large areas of Canada are covered by deposits of this type. The early glacial lakes covering large parts of Alberta, between the front of the Keewatin ice sheet and the Rockies, supplied the silty soil worked around Edmonton.

The clays of the glacial lake, Agassiz, cover wide areas in Manitoba, as do these clays of Lake Ojibwa in northern Ontario.

In northern New Jersey similar clays occur, having been deposited in ponds by streams flowing from the front of the retreating ice sheet. They are often fine grained and laminated, and of good plasticity.

The impure plastic clays in swamps along the larger streams and near the coast in the Coastal Plain of Georgia belong to this division of clays. They are very fine grained, plastic and contain organic matter.

In southern New Jersey also occur large deposits of estuarine clays. They were laid down in sheltered bays and estuaries. They are fine grained and interbanded with sand layers.

The extensively worked clays of the Hudson Valley are of this type, the section usually being made up of an upper sand bed, a yellow weathered clay, and a blue clay. The clays are laminated, plastic and red burning.

All these clay types are simply mixtures of mineral matter, crystalline and colloidal gel, and their plastic properties depend on the relative amounts and specific properties of these two. They have all suffered water action for long periods of time.

By far the most important variety of transported clay is that which was deposited under marine conditions. The marine clays are made

up of the finest products of rock decay and transportation. The material was brought down to the ocean by the streams and deposited in quiet water, the finest material farthest from shore. Such formations cover wide areas and are known to be often of great thickness. They occur widespread in beds of Silurian, Devonian, and Carboniferous age. Those of the Mesozoic are also quite persistent.

Many marine clays, since their formation, have been covered by other materials and often are consolidated enough to be called shales. It is well known that where consolidation is due to pressure alone, the plastic properties are little affected, but where there has been a change in the mineral matter and recrystallization has gone on, the plasticity is lowered. Weathering or fine grinding and pugging has then to be resorted to in using the material in a wet or stiff-mud process.

Marine clays vary in mineral and chemical composition and in physical properties, as do the clays of other types, depending on the composition of the rocks of the drainage basins from which the material was derived.

In the Coastal Plain areas of the Atlantic States occur plastic kaolins and fire clays derived from the residual clays of the highly feldspathic crystalline rocks of the Piedmont Plateau. They were transported only short distances, yet are much more plastic than the residuals from which they were derived.

Other examples could be given of clays, mixtures from different drainage basins, and from basins of basic rocks, but the increase in plasticity in the case of the transported kaolins is most striking.

Wind-Formed Deposits.—In the Mississippi Valley region are found extensive deposits of a silty, often calcareous, clay called loess. It is much used for making common wet-mud and dry-press brick.

The loess clays are of post-glacial age and represent collections of wind-blown material. In Missouri the deposits are of great thickness.⁸⁸ They are usually non-stratified and exhibit a columnar structure in the banks. The material is usually fine grained, the particles varying in size from 0.1 mm. to ultra-microscopic dimensions. Mixed with water it develops low plasticity and is often hard to dry safely.

In the light of the reasoning in this paper, low plasticity should be expected in wind-formed clay. As in the other types, the degree of plasticity depends on the past history of the material. The large quantities of calcareous drift gathered by the glaciers from the sedimentary rocks to the north, and surrounding the Mississippi Valley, furnished the material of the loess. The only period during which it was subjected to water action was that of the life of the ice sheet. Following its deposition as drift in front of the retreating glacier, strong winds worked

⁸⁸ Wheeler: *Clay Deposits*, Missouri Geological Survey, vol. xi, p. 483 (1896).

it over, carrying away the finest particles to form the loess. Surficial weathering has since taken place to some extent and has increased the plasticity locally.

V. SUMMARY

In review, the following points may be emphasized:

(1) Clean crystalline rock powders do not develop plasticity on the addition of water, no matter how fine they may be ground in the dry form.

(2) Such powders if ground in water for a long time do develop some plasticity.

(3) Those powders containing minerals least easily attacked by water, such as quartz and feldspar, develop least plasticity.

(4) Water exerts a solvent action on the mineral grains, resulting in colloid and true solution, the colloid modifications remaining as coagulated sols, gelatinous coatings over the mineral grains.

(5) Organic gel bodies in small amounts increase the viscosity of the water films in mineral powders. The effect seems to be more pronounced than that of equal amounts of the inorganic gels, as shown by Grout's experiments with agar and with gelatinous alumina.

(6) The amount of gel matter present in a clay depends on the past history of the deposit. Clays are simply the weathering products of crystalline rocks in which the most soluble constituents have been removed, leaving behind more resistant secondary products (crystalline and amorphous) with gelatinous coatings.

(7) The most highly plastic clays are those that have been subjected to the action of water for long periods. Residual kaolins are of low plasticity, but water-transported kaolins of exactly the same chemical composition are quite plastic.

VI. CONCLUSION

Plasticity in clays, then, is due to the gelatinous state of matter, a state common to them because of their mode of origin. This gelatinous matter may be silicic acid gel, alumina gel, iron oxide gels, silicate gels, or organic gels. Two or more of these are usually present, and their effect will be further modified by adsorbed salts and the relative proportions of large and small grains, and to a limited extent by the shape of the grains.

The particular kind and amount of gelatinous matter present, the size and shape of grain, and the relative proportions of large and small grains, are important factors in determining the other related physical properties of tensile strength and air shrinkage.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Some Defects of the United States Mining Law

BY COURTENAY DE KALE, TUCSON, ARIZ.

(New York Meeting, February, 1915)

REVISION of the United States mining law is needed chiefly because of the following reasons:

1. The conceptions as to the characteristics of orebodies that were held at the time the statute of 1872 was enacted have since proved to be in many respects erroneous.
2. The operation of the existing law has been found to work unnecessary hardship upon many claimants to mineral land.
3. The existing statute fails in large degree to encourage development of mineral resources.
4. The existing law often produces conflict between placer and lode-claim rights, working unnecessary injustice to placer claimants.
5. The methods of establishing the rights of locators on United States mineral land are defective, and are not uniform throughout the States within which such lands exist.

LAW OF THE APEX

The objections to the so-called "law of the apex" are so well known as to require no detailed statement. It is sufficient to point out that the apex rule was based upon the notion that ore deposits consisted of well-formed veins included between definite walls, like the slice of ham in a sandwich. The fact is that veins conforming to that simple type are rare. Irregularity is the rule, and the chief metal production of the United States comes from masses which bear no resemblance to the tabular form of vein. Manifestly the law of the apex fails completely in such cases. Moreover, the tendency is constantly toward the exploitation of ores of lower grade than could formerly be worked, and such deposits exist mainly in the form of large irregular impregnated zones and masses, so that the future development of our mineral resources would be continuously embarrassed by shadowy claims to extra-lateral rights by the locators of so-called apexes.

INITIATION OF MINERAL RIGHTS

The spirit and intent of the United States mining statute possess the merit of liberality toward the poor prospector, in which respect our law is superior to that of most other countries. This spirit must be jealously maintained, and all efforts to enact a rich man's law must be resisted. The advantage to the nation through the actual development and exploitation of our mineral resources is vastly greater than the benefits represented by the sums that might be paid into the Federal treasury in the form of purchase price of claims, or as rental under a leasehold system. The development of industry means the broadening of opportunity for many workers and their families, provides room at home for an increasing population, and involves income to the government through taxation so large in the aggregate as to dwarf utterly the revenue that might be derived from the sale or rental of mineral lands. The leasehold system is peculiarly objectionable from this point of view, since it necessarily regards the mineral resources as a basis of direct taxation, payable as a condition of tenure, and this leads to a restriction of the right of prospectors to extract and ship metalliferous products without prepayment of the rental. It therefore imposes a burden upon locators from the beginning of their work, seriously limiting the opportunity for poor men to prove the value of their claims. The law should in every possible way promote the discovery and exploitation of the mineral resources, and should never prove a deterrent.

Under the existing statute the locator of a claim is supposed to have made an actual discovery of valuable mineral; he is required to so state in his location notice; and, very logically, following the plain language of the statute, he is required to demonstrate such discovery in the case of a claim located in a National Forest before acquiring any rights to operate thereon. As a matter of fact, the actual discovery of valuable mineral on the surface is rare. Even in past decades the prospector has usually been led to locate a mining claim on the strength of recognized indications which could be expected to lead to a valuable deposit only after the performance of a considerable amount of costly exploratory and development work. To-day the dependence of the prospector is more than ever upon indications, since the richer deposits, yielding workable ore at the surface, have been mostly discovered. We are entering a new era of the mining industry, in which reliance upon geologic indications must be increasingly great. The prospector is no longer actuated by hopes of striking a bonanza with his toe as he wanders over the hills. He has absorbed a certain amount of geological information from the army of trained men who are exploiting the mines of the country, and he possesses a keener appreciation of the salient features of ore formation and of the conditions that warrant belief in more deeply seated deposits. To require a

declaration of actual discovery is, more often than not, to turn the locator of a claim into a perjurer. The law should frankly provide for valid location based upon evidence which to the best knowledge and belief of the locator gives promise of leading to the development of a valuable deposit. He should then be accorded ample time within which to prove his claim.

THE MINING DISTRICT

The statute of 1872, following the customs of the early miners of the West, provides for the organization of mining districts, with power to create certain regulations, and to record locations. This primitive system served a useful purpose when the West was young and politically unorganized, but to-day it constitutes an anomaly, utterly superfluous, and fraught with dangerous possibilities. The great majority of mining districts exist to-day in name only, and for the most part are wholly disregarded by locators. Nevertheless, in many States no laws have been enacted, or upheld by the courts of appeal where they had been enacted, to provide for securing valid claims without their intervention, and such mining districts as are still kept alive by active organizations are mostly maintained by and in the interest of rascals for the purpose of defrauding men who make important discoveries within their limits. It is no longer difficult to reach the county seat to record location notices, nor to ascertain the State laws governing validity of locations, but it is often an almost hopeless task to find the district recorder, or even to find out whether there is such an official, and it is a still harder task to make sure that the regulations painfully dug out of musty old notebooks which had lain amid rubbish in the corner of a cabin in the hills, actually constitute all the regulations which the district organization may have enacted at various periods of its history. There are many crafty scoundrels, who might have made daring bandits a generation ago, who practice the safer villainy to-day of dominating a mining district, like a Spanish *cacique*, lying in wait to trap the unwary prospector or development company by artifices which the present mining law renders possible. That which was a beneficent provision under frontier conditions has become the tool of crooks in a civilized society. The mining district has served its function, and should now be abolished.

TITLES

The American is wedded to land tenure by title. The sentiment in favor of title and against possession under leasehold is fully warranted. A title once held, though forfeit to the State for delinquent taxes, still clouds any future title that may issue, until a court title has been obtained in an action brought against the former owner, his heirs, or assigns. This means that, under conditions which might render it impossible for a

man to comply with all the requirements for retaining a leasehold, the owner of a mining title in fee simple has ample time to recover from his disabilities, whether of war, of poverty, or sickness, make good his delinquencies, and finally, if need be, throw his case upon the mercies of a court of equity. The difference between tenure by title and tenure by leasehold may be illustrated by the homely simile of a man hanging in a shaft by a slender rope which has just the requisite strength to sustain his weight. If a single strand is worn through on the rocky edge of the shaft mouth, he falls; but if he were hanging by a rope strong enough to sustain many times his weight one strand after another might be cut through and the man could still hang on. The mining law is for both the rich and the poor, but it should not be strong enough for the rich and too weak for the poor. The prospector who has more hope than cash wants a rope strong enough to hold fast to that hope through all the storms of adversity.

It has been pointed out that tenure by assessment work is open to grave abuse, as the law stands. Not only do men credit themselves with larger wages than they could possibly earn if working for others, so as to swell the account quickly to the \$100 limit, but they falsify, and they do work that is easy, just to hold the claim, rather than work that is effective in the development of the alleged mineral deposit. The purpose of the law is and should constantly be toward the stimulus of actual development. Its two functions are to stimulate and to protect. The present statute has unfortunately lent itself in this particular to the retardation of industry, to the purposes of the "dog in the manger," and to those of the grafter. This is an evil that should be cured. Every operator knows how persistently clever schemers locate claims around a promising mine without hope of finding ore, but with the expectation that some time the expanding operations of the successful company will need the ground for a tailing dump, or a millsite, or for its growing camp. There is no limit to the length of time that a claim may be held by the mere performance of assessment work. The law requires that \$500 worth of work be done before patent may issue. That is equivalent to five years' assessment work. It is often open to question what may or may not constitute \$500 worth of work. This depends on many factors; but there can be less question about the granting of title upon the demonstration of the existence on the claim of commercially valuable mineral. If the claimant does not find commercially valuable mineral during a period of five or six years it is fair to assume that he never will find it, and such claimant should be incapacitated from securing patent to the same ground by relocation, or as a result of assignment to him from other locators, for a period longer by one year than the time within which a locator must proceed to patent or abandon.

It will be seen that the foregoing is distinctly in line with two objects

of a proper mining statute: viz., to foment the mineral industry, and to prevent the use of the law for purposes of graft. There should further be some provision for determining whether abandoned claims are to be again open to location, or whether they should be classified under another heading and offered for sale as public land. If so offered, the price should be enough higher than that required, after the demonstration of valuable mineral, for the acquisition of a mining title, as to prove an incentive to acquire by development of mineral rather than by purchase as open public land.

PLACER VS. LODE CLAIMS

The existing statute requires that placer claims be located in conformity with the land divisions where the area has been surveyed; that is, they must be described as, for example, the N. W. 1/4 of section so-and-so, or the N. 1/2 of the S. W. 1/4 of the S. W. 1/4 of section so-and-so, of township and range as per this or that meridian. Theoretically the system is admirable. If the land is surveyed, describe it thus, on the co-ordinate principle, whereby any one may put his finger on the precise square upon a map which represents the claim in question. Successful mining, however, depends not alone on theory; it depends very much indeed upon the practical hard facts one clashes with in the field. The hard fact in this case is that in the more rugged and barren parts of the West, where mines have an ungovernable propensity for secreting themselves, the population is scanty, the ranches are confined to widely scattered *arroyos* where a few acres of arable land exist, and where the conflict of neighbors has not led to checking out the surveys. In large part the survey contracts in the remoter regions were let with an eye to political advantage, and were treated by the contractors as a private graft. Here and there an area will be found that was honestly surveyed, where the corners were actually set, even far back from the more frequented trails and roadways, where witness monuments were really erected or witness trees blazed and inscribed; but these are the exceptions. Usually the corners were set only where failure to do so might be readily detected. Hence the discoverer of a placer deposit might hunt in vain for any sign or mark to guide him to a known co-ordinate. In order to ascertain his position on the map he must either run out the survey himself from some corner stone many miles away, or at large expense hire a surveyor and his corps of assistants to do it for him. The law never contemplated throwing such an unjust burden upon the locators of placer claims, but that is precisely the disability under which they labor. In revising our mining law we must do so in the light of facts. When a locator cannot, with a reasonable amount of effort, ascertain his co-ordinate position according to the survey of record, he should be permitted to claim and hold in accord with the actual area and position of his placer as referred

to easily recognizable landmarks. Even though it result in cutting land on the bias, and in playing tricks with the chessboard map in the Surveyor General's office, this disturbance of the idealistic harmony would be no greater than that produced by the old confirmed Spanish grants which frightfully disfigure Uncle Sam's co-ordinate symmetry in every part of the Southwest; nor would it deface the maps or upset the calculations of locators any worse than the lopsided "lots" which surveyors have been permitted to mark off as fillers along the township lines where the surveys refused to close. These examples show how vain it is to expect perfection, and since the co-ordinate system of land survey has in reality been chopped into a crazy quilt by force of circumstances, and as vast areas have been reported as surveyed which have never seen either compass or transit, it is wholly unwarranted to demand that the locator of a placer claim should pretend to define the position of his claim in the way that the present law requires.

Furthermore, it is quite irrational to require, as results under the statute, that a locator take up areas lying on high ground, of no possible value for placer mining, just because they happen to fall within the subdivisions of a section needed to include the valuable bottoms. It withdraws land which might better be left open for other uses, and it imposes upon the placer miner the burden of buying extensive acreage which he cannot use, when he proceeds to patent. This means, therefore, that common sense demands, even in the case of association claims, the much-maligned "shoe-string" form of placer location as the one that fits the facts of nature and of an imperfectly realized public-land survey.

Another evil to be cured in revision of the mining law is the absurd conflict between the lode and the placer miner. At present a placer locator can obtain no prior rights as against a subsequent locator of a cross lode. Here again the dual purpose of the law, to foment and to protect, should be kept in view. A placer mine by its very nature cannot be expected to have a long life, while a lode mine may continue producing for many generations. There is consequently no reason why a placer title should carry rights to mineral in place beneath the alluvial deposit. In many of the Eastern States, following the ancient theory that the metals belong to the sovereign, land titles do not carry an exclusive right in metalliferous veins which may be discovered at some future time. Precedent for such a distinction is abundant. In this view a lode title could be issued subject to pre-existing superficial rights obtained by placer title, terminating, so far as mineral rights are concerned, at rock in place. No further interference as a result of debris, right of way, water rights, and the like, could be anticipated than that which occurs under the conditions contemplated by the existing law, which admits of placer claims adjoining both side lines of a lode claim 600 ft. wide. It certainly is a manifest injustice to a placer locator that he be made subject to an ad-

verse lode location of later date, which may cut the placer claim in the middle, and seriously hinder its proper economic exploitation, or wholly preclude its operation. The common practice of locating supposed lode claims across a placer, for purposes of blackmail, is in itself a severe criticism upon the provisions of the existing statute, and is a sufficient warrant for careful revision in this respect. As matters now stand the locators of placer claims who hope to operate undisturbed by litigation from later lode claimants are forced to locate every cross seam, or prominent ledge of rock, or faulted zone, as lode claims, in order to protect their placer claims, thus incurring a great deal of unnecessary expense. That which embarrasses and heightens the cost of production is not in line with the theory of the conservation of natural resources.

UNIVERSALITY OF THE LAW

Out of the same theory that provided for the establishment of local mining districts comes the opportunity for the several States to further complicate the mining law by legislation concerning the details of location, proof work, record, and other details. The failure of the Federal statute to settle all these matters was due to the frankly confessed ignorance of the majority of the representatives in the Congress, concerning mines, and ore deposits in particular, and concerning Western conditions in general. The congressional debates at the time, especially those in the Senate, where the bill was the subject of very earnest consideration, show the helplessness of the Congress in the face of these matters. There was then no Geological Survey with a vast store of facts and a corps of trained experts to guide and counsel; there was no Bureau of Mines to assist with testimony as to actual mining conditions; there were no trains of Pullman cars to facilitate the acquaintance of the Easterner with the West. Except for the good sense of the late Senator Stewart of Nevada, whose fund of practical information and forcefulness of character enabled him to dominate his colleagues, the law would not have been sound enough to have stood the test of 42 years as a fairly satisfactory instrument for the protection of an industry developing out of the relatively small beginnings of a frontier epoch into the mammoth enterprises which flourish under it to-day. The success of the mining law compels profound admiration and respect. Its deficiencies, however, as revealed by changing conditions and expanding knowledge, call for revision; and in curing the evils which more directly affect tenure in mineral land, it is also important to provide for those things which come under the head of "regulations," and which have hitherto fallen within the purview of mining districts and State legislatures, to the end that uniformity in law and procedure touching mining claims may exist wherever there remain public mineral lands open to location.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Method for the Determination of Gold and Silver in Cyanide Solutions

BY L. W. BAHNEY,* NEW HAVEN, CONN.

(New York Meeting, February, 1915)

MANY methods for the determination of gold or silver, or both, in cyanide solutions have been published, which with care in manipulation, and modification in some cases, will give results that are satisfactory. It is possible to classify or group these methods as follows:

1. Evaporating the solution in a porcelain or agate-ware dish containing litharge,¹ or in a "boat" made of lead foil.
2. Forming a lead sponge containing the precious metals by means of zinc shavings,² zinc dust,³ or a piece of aluminum.⁴
3. Decomposing the cyanide solution with an acid and precipitating the precious metals by the use of one or a combination of some of the following: Copper sulphate,⁵ sodium sulphite,⁶ hydrogen sulphide,⁷ cement copper.⁸
4. Precipitating the silver by zinc dust held in a Gooch crucible and determination with a standard solution of sulphocyanate.⁹
5. Electrolysis.¹⁰
6. Colorimetry.¹¹

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¹ T. Lane Carter: *Engineering and Mining Journal*, vol. lxxiv, No. 20, p. 647 (Nov. 15, 1902).

² Alfred Chiddy: *Engineering and Mining Journal*, vol. lxxv, No. 13, p. 473 (Mar. 28, 1903).

³ W. Magenau: *Mining and Scientific Press*, vol. xcii, No. 15, p. 259 (Apr. 14, 1906).
N. Stines: vol. xcii, No. 17, p. 278 (Apr. 28, 1906).

H. L. Durant: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iii, pp. 105 to 111 (1902-03).

⁴ T. P. Holt: *Mining and Scientific Press*, vol. c, No. 24, p. 863 (June 11, 1910).

⁵ A. Whitby: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iii, p. 6 (1902-03).

⁶ J. E. Clennel: *Cyanide Handbook*, p. 443 (1910).

⁷ Henry Watson: *Engineering and Mining Journal*, vol. lxxvi, No. 26, p. 753 (Dec. 24, 1898).

⁸ Albert Arents: *Trans.*, xxxiv, p. 184 (1904).

⁹ G. H. Clevenger: *Engineering and Mining Journal*, vol. xciv, No. 18, p. 892 (May 3, 1913).

¹⁰ Miller: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, May 16, 1905.

¹¹ James Moir: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 298 (1903-04).

A. Prester: *Proceedings of the Chemical, Metallurgical and Mining Society of South Africa*, vol. iv, p. 385 (1903-04).

Eliminating Group 4 because of its applicability to silver solutions only; Group 5, because of the time and apparatus required; and Group 6, because of the skill required, and the difficulty of maintaining the standards; which method of the remaining groups will give accurate results in the shortest time?

In the Hammond Laboratory of the Sheffield Scientific School, where many ores are tested for treatment by the cyanide process, the resulting solutions will cover a wide range, when their contents of base metal compounds are considered, and it is in the laboratory work just as much as in mill work that a reliable method that will not require too much time is needed. This is especially important in teaching.

What criticism I have to make has been brought about by doing what every operator does—trying the various methods to find the one that will give “good results.”

Group 1 requires too much time; a large hot-plate surface if many determinations are to be made; scraping of the dishes clean to remove all particles; breaking up the mass; fluxing and fusing in a crucible. Evaporating in a lead boat is uncertain, because some lead foil may be quite thin and perhaps pitted, so that the solution will leak through as the evaporation proceeds and the cyanide solution becomes concentrated.

Group 2 includes the method suggested by Alfred Chiddy and others that are modifications of it.

The idea of the formation of a lead sponge to contain the gold and silver as suggested by Chiddy is a clever one, and it appealed to every one having anything to do with cyanide solution. To be able to remove from the dish a small sponge of lead that could be cupelled was a great advancement in the work. It is difficult to get good results if it is followed as printed,² so that its modification as suggested by Stines, Magenau, Holt and others is a natural outcome. Any of the methods of this group that will give a sponge of closely cohering lead, containing all the gold and silver, in a reasonable time, is a “good one;” but when the sponge breaks into small pieces they must be collected in some manner and filtration is the next step.

When the lead has been collected on a filter paper it then becomes necessary to scorify or dry the paper and reduce it in a crucible with the necessary fluxes.

It has seemed to me that it is right here that an opportunity exists for a new method, either for the formation of a good sponge or to save the broken sponge formed by any of the other methods, and at the same time eliminate scorification.

Group 3 includes all those methods that permit the use of a large quantity of solution and from which the precious metals may be precipitated as mentioned above. Whether it is necessary to use so large a quantity, aside from experimental work perhaps, I shall leave to the individual

operator. My own objections to this group are: The quantity of solution involved; the time required; and the necessity of filtration and scorification.

In order to present this new method clearly I have numbered each successive step in making the determination and have included the photograph to show apparatus, etc.

I. Procedure for New Method

1. Into a 250-c.c. beaker (No. 2 low form) pour 5 assay tons (146 c.c.) of cyanide solution.

2. Add 20 c.c. of a 20 per cent. solution of lead acetate.

3. Add 15 c.c. of concentrated hydrochloric acid.

4. Place a $\frac{1}{4}$ -in. rod of zinc in the beaker (No. 1 in the photograph).

5. Place beaker on a hot plate—bumping does no harm if it will not break the beaker by raising it off the plate.

6. As soon as the solution boils, leave it so for 5 min.; then remove from hot plate.

7. Fill with cold water; then decant about half and again fill with cold water.

8. Twist the zinc rod quickly between the finger and thumb and draw it out of the sponge.

9. If any small particles of lead are left adhering to the rod at about where the top of the solution touched it, draw the sponge up the side of the beaker with a glass rod.

10. Touch the zinc rod to the sponge to free the particles.

11. Wash the sponge three or four times with cold water to remove zinc chloride.

12. Press it against the side of the beaker with a glass rod to remove the water.

13. Decant the water and wash again.

14. Place the dewatered sponge on a piece of sheet lead 2 in. square, then fold it. It is now ready for cupellation.

II. Procedure When the Sponge Breaks

When the sponge breaks into large pieces it is possible to unite them by pressing together with a glass rod. If there are many small particles it may not be possible to unite them by pressing together. Then this method is suggested:

15. Fit a 2-in. funnel into a filtering flask.

16. Cut a piece of sheet lead 3 in. square; fold as you would a filter paper.

17. Cut off the corners; then open it to fit the funnel.

18. Place about 5 g. of test lead in the lead cone and push it down with the finger, then smooth out the folds and creases so that it will fit well.

19. Lift it from the funnel and prick seven or eight holes in the apex or point by means of a pin, then put it back into the funnel.

20. Complete 9 and 10; then start the filter pump.

21. Tip the beaker and draw most of the lead sponge into the lead cone by means of a glass rod (without a rubber tip).

22. Pour the remainder of the solution into the cone, rinse beaker and wash contents of cone three or four times.



FIG. 1.—APPARATUS USED IN THE NEW METHOD FOR THE DETERMINATION OF GOLD AND SILVER IN CYANIDE SOLUTIONS.

23. Tamp the lead down with the flat knob of a glass rod. (See 4 in Fig. 1.)

24. Stop suction, remove cone from funnel and fold it in carefully. (See 5 in Fig. 1.) It is now ready for cupellation.

25. Draw a hot cupel to the front of the muffle and place the cone in it. (See 6 in Fig. 1.) When the water has been driven out, push the cupel into a hotter part of the muffle and finish the cupellation in the regular manner.

III. *Table Showing Amounts of Solution of Lead Acetate and Acid Necessary for Different Quantities of Solution*

Assay Tons	Equivalent in c.c.	Lead Acetate Solution, c.c.	Conc. HCL, a.c.	Time (including 5 min. boiling), Min.
5	146	20	15	20
10	292	25	25	34
15	437.5	30	40	35
20	583	45	50	42
25	729	50	75	44

IV. *Notes and Comment*

The generation of hydrogen along the zinc rod is sufficiently active to prevent the adherence of the lead. A strip of aluminum does not work so well.

The method has been used in the assay of a number of solutions containing a variety of base-metal compounds and in each case the sponge remained whole. With solutions from cobalt ores that contain much silver the sponge is apt to break.

The amount of time as given in the table will vary with the heat of the hot plate—those given are averages.

In order to keep the weight of the lead to be cupelled down to a minimum, thin sheet lead should be used.

A cone as described in 16 and 17 will correspond to a filter paper $7\frac{1}{2}$ cm. in diameter and, made of heavy sheet lead, will weigh about 7 g.

The pin holes should always be as near the point as possible.

A screen analysis of the test lead used in developing this method is as follows: + 30, 7.0 per cent.; + 60, 28.6; + 100, 23.2; - 100, 41.2 per cent.

It is quite possible to have a cone manufactured as are bottle caps. This would lessen the weight about 50 per cent.

V. *Scorifying the Precipitate*

The following method may be used for scorifying the precipitate obtained by any of the methods of Group 4:

26. Collect the precipitate in a 9-cm. filter paper; wash it down into the point.

27. Make a cone from a disk of sheet lead $3\frac{1}{2}$ or 4 in. in diameter.

28. Punch 10 or 12 holes at the point.

29. Fold the filter paper into a small wad and place it in the point of the cone.

30. Pour on top of the paper 10 g. of test lead.

31. Fold the lead cone so as to include its contents and place it in a glazed scorifier at the mouth of the muffle.

32. When the paper has become dry and begins to char, the gases will burn as they come from the pin holes. As soon as the flame ceases place the scorifier in a hotter part of the muffle. The lead cone will hold the paper firmly while it is burning; so there is no danger of its unfolding and scattering the precipitate.

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Barite of the Appalachian States

BY THOMAS L. WATSON AND J. SHARSHALL GRASTY, UNIVERSITY OF VIRGINIA,
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(New York Meeting, February, 1915)

INTRODUCTION

THE users of barite in the United States derive their supply partly from the domestic production and partly from the imports from foreign countries. According to the Mineral Resource division of the U. S. Geological Survey, the domestic production of barite in 1913 was 45,298 short tons, valued at \$156,275, or \$3.45 a ton, and the imports of crude barite were 35,840 short tons, valued at \$61,409, or \$1.71 a ton. These figures clearly show a decided advantage in cost of the foreign over the domestic barite to users in the United States. The difference in cost is the only advantage to the consumer of foreign barite, for there are large reserves of the mineral in the United States of good grade. The production of many of the operating mines is capable of decided increase, and it is believed that the operating mines together with the large reserves of undeveloped deposits of barite are entirely adequate to meet the demands of the users in the United States.

Deposits of barite are known in California, Idaho, Nevada, and Alaska, but most of them are undeveloped because there is, apparently, no market for the material in that region. The domestic production is derived from Missouri and the Appalachian States, the greater part being obtained from Missouri.

Following the outbreak of war in Europe many users of foreign barite in the United States have been forced to seek their supplies at home. It seems opportune at this time, therefore, that a general review be given of the barite industry in the Appalachian States, one of the two areas from which the domestic supply of the mineral is derived, and that attention be directed to the undeveloped deposits in the region. Briefly, then, this paper reviews the barite industry in the Appalachian States, and describes the occurrence, preparation, and uses of the mineral.

GENERAL CHARACTER

Barite, known commercially as barytes or heavy spar, is barium sulphate (BaSO_4), containing when pure BaO, 65.7 per cent., and SO_3 , 34.3 per cent. The specific gravity is 4.3 to 4.6, and the hardness 2.5

to 3.5. The hardness is about that of calcite, but barite can be readily distinguished from calcite by its greater specific gravity and by not effervescing with acids. The mineral is usually white, but it is often found stained red, pink, or yellow by iron oxide. It is usually crystalline, opaque to translucent, and occurs in a variety of forms, commonly as aggregates of straight and slightly curved cleavable plates, as granular, fibrous, and earthy masses, and as single and clustered crystals. It is rarely found pure in workable deposits, the common impurities being silica, lime, magnesia, and the oxides of iron and aluminum. In many deposits galena and fluorite are associated with the mineral. The minable material will usually carry from 95 to 98 per cent. barium sulphate.

PRODUCTION¹

During the 10-year period 1904 to 1913, inclusive, the production of barite has been entirely from the following States, arranged alphabetically and without reference to order of production: Alabama, Georgia, Kentucky, Missouri, North Carolina, South Carolina, Tennessee, and Virginia. Not all of these, however, have produced during the same year. In 1913, the production came from Missouri, North Carolina, Virginia, Georgia, and South Carolina. The Kentucky deposits did not produce in 1912 or 1913.

The following table gives the total production of crude barite in the Appalachian States and in the United States in short tons from 1904 to 1913, inclusive:

Production of Barite

	Total Production, ^a Appalachian States		Total Production, ^b United States	
	Quantity	Value	Quantity	Value
1904.....	40,229	\$99,406	65,727	\$174,958
1905.....	21,474	64,705	48,235	148,803
1906.....	21,362	66,888	50,231	160,367
1907.....	45,582	129,318	89,621	291,777
1908.....	22,208	63,674	38,527	120,442
1909.....	27,130	89,919	61,945	209,737
1910.....	19,997	46,148	42,975	121,746
1911.....	16,945	41,412	38,445	122,792
1912.....	12,948	36,278	37,478	153,313
1913.....	14,167	38,637	45,298	156,275

^a Includes Virginia, North Carolina, South Carolina, Georgia, Tennessee, Kentucky, and Alabama.

^b Includes all States under (a) and Missouri.

¹ Unless otherwise stated all statistical data given in this paper are taken from *Mineral Resources of the United States*, U. S. Geological Survey.

Average Price per Ton

	Missouri	North Carolina	Tennessee	Virginia	Other States	Total
1905..	\$3.14	\$3.90	\$1.62	\$4.30		\$3.08
1906..	3.24		1.67	3.85	\$2.94 ^a	3.19
1907..	3.09	3.76	1.78	3.55	4.18 ^a	3.28
1908..	3.48		1.43		3.51 ^b	3.13
1909..	3.44				3.31 ^c	3.39
1910..	3.29		1.54		2.55 ^d	2.83
1911..	3.79		2.27 ^e		2.63 ^f	3.19
1912..	4.77		2.34 ^e		2.99 ^g	4.09
1913..	3.78		1.70		2.91 ^h	3.45

^a Includes Alabama, Kentucky, Georgia.

^b Includes Georgia, North Carolina, and Virginia; Kentucky was \$4.11.

^c Includes Georgia, Kentucky, North Carolina, Tennessee, and Virginia.

^d Includes Georgia, Kentucky, North Carolina, and Virginia.

^e Includes Tennessee and Kentucky.

^f Includes Georgia, North Carolina, South Carolina, and Virginia.

^g Includes Georgia, North Carolina, and Virginia.

^h Includes Georgia, North Carolina, South Carolina, and Virginia.

IMPORTS

The subjoined table gives the imports of barite for consumption during the last 10 years, from 1904 to 1913, inclusive.

Barite Imported into the United States, 1904 to 1913, in Short Tons

	Manufactured		Unmanufactured		Total Value
	Quantity	Value	Quantity	Value	
1904.....	6,630	\$48,658	7,492	\$27,363	\$76,021
1905.....	4,803	39,803	14,256	62,459	101,262
1906.....	4,807	37,296	9,190	27,584	64,880
1907.....	11,207	96,542	20,544	76,883	173,425
1908.....	3,401	29,168	13,661	58,822	87,990
1909.....	3,016	25,679	11,647	29,028	54,707
1910.....	3,565	29,782	21,270	48,457	78,249
1911.....	3,147	22,083	20,214	36,643	58,726
1912.....	3,679	26,848	26,186	52,467	79,315
1913.....	5,463	38,155	35,840	61,409	99,564

The total yearly value of the imports of barium compounds, 1904 to 1913, including barium carbonate (witherite), barium binocide, barium chloride, and blanc fixe or artificial barium sulphate, follows:

Total Value of Imports of Barium Compounds, 1904 to 1913

Year	Value
1904.....	\$242,804
1905.....	257,427
1906.....	335,011
1907.....	357,117
1908.....	319,114
1909.....	399,376
1910.....	470,449
1911.....	398,213
1912.....	376,017
1913.....	391,470

The mineral witherite (barium carbonate) is admitted free of duty but under the Underwood tariff the manufactured barium carbonate is dutiable at 15 per cent. *ad valorem*, the natural barite at 15 per cent. *ad valorem*, the artificial barium sulphate or blanc fixe at 20 per cent. *ad valorem*, barium binoxide at $1\frac{1}{2}$ c. a pound, and barium chloride at $\frac{1}{4}$ c. a pound. On raw barite the duty is \$1.50 per ton, on prepared or manufactured barite \$5.25 per ton, and on blanc fixe or artificially prepared barium sulphate the duty is $\frac{1}{2}$ c. per pound.

GEOGRAPHIC AND GEOLOGIC DISTRIBUTION

Barite has rather wide geographic distribution in the Appalachian States and occurrences of the mineral are known in each of the major physiographic divisions of the region: (1) The Piedmont Plateau, (2) the Appalachian Mountains, (3) the Appalachian Valley, (4) the Cumberland Plateaus, and (5) the Interior Lowlands. (See map, Fig. 1.) It is found in workable quantity in many localities within the Appalachian States, but in recent years the sources of supply have been from Virginia, North Carolina, South Carolina, Georgia, Tennessee, and Kentucky. Deposits of the mineral are also known in Alabama, Maryland, and Pennsylvania. Practically the entire domestic production of the mineral comes from Missouri and the Appalachian States, the former supplying about 65 per cent. of the total production.

Geologically, the barite deposits of the Appalachian States are associated with rocks which range from pre-Cambrian to Triassic in age, but the exact age of the deposits themselves is unknown.

The age of the rocks in which the barite deposits are found may be tabulated as follows:

1. Triassic: Virginia.
2. Carboniferous (Mississippian): Western Kentucky.
3. Cambro-Ordovician: Central Kentucky, Tennessee, Appalachian Valley region of Virginia, Georgia, Alabama, Maryland, and Pennsylvania.
4. Pre-Cambrian and Cambrian: Piedmont Plateau province of Virginia and North Carolina, including the Madison County deposits in the western part of the State.

Of these four horizons, the Cambro-Ordovician is vastly the most important in the Appalachian States and to it belong the deposits of Missouri, the principal producing area in the United States. The deposits of barite concentrated in the residual clays derived by weathering from the rocks of the several horizons tabulated above are later in age, probably much later.

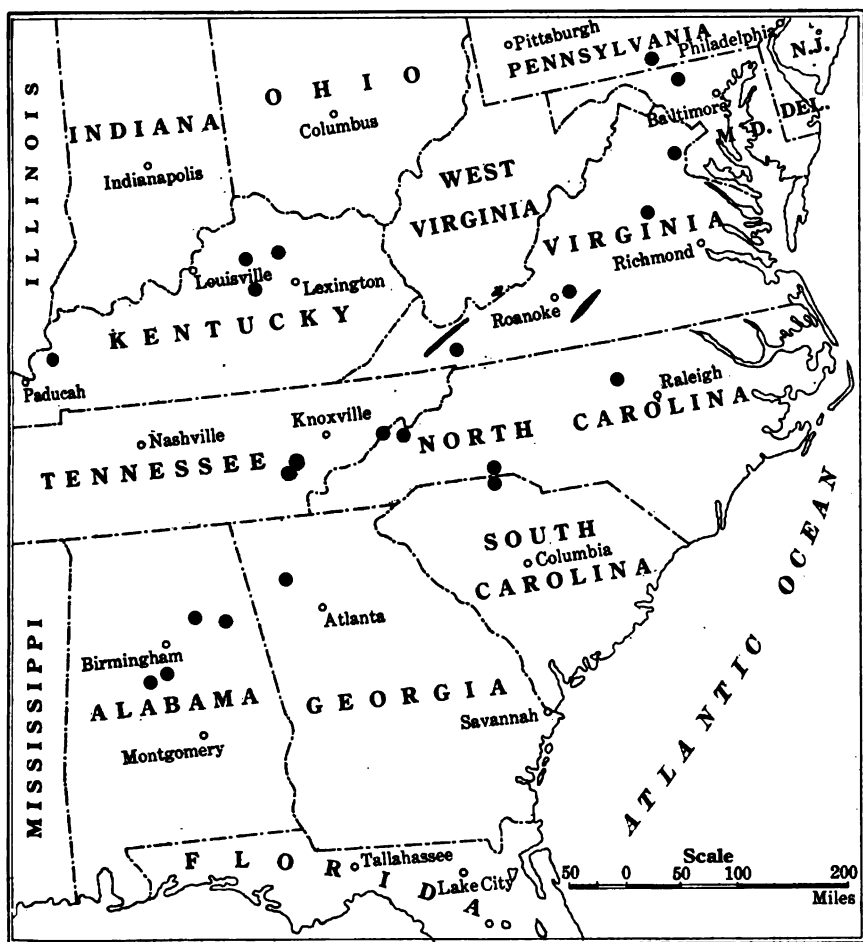


FIG. 1.—MAP SHOWING BARITE DEPOSITS IN THE APPALACHIAN STATES.

MODE OF OCCURRENCE

Barite is not a pyrogenetic mineral, nor is it known in contact-metamorphic deposits, in pegmatite veins or dikes, or as a product of dynamo-regional metamorphism. It is a common mineral, however, in many kinds of rocks—igneous, sedimentary, and metamorphic; and in nearly all

if not in all cases it has formed by deposition from aqueous solutions. Many writers record the deposition of barite from spring and mine waters of carbonate and even sulphate composition, but especially sodium chloride waters.² Depositions from both hot and cold water are recorded.

Thus far, therefore, the occurrence of barite is limited to deposits of moderate and shallow depth, either nearby, sometimes in, or removed from igneous rocks, but not formed, according to Emmons,³ in deposits of the deep-vein zone.

The occurrences of barite in deposits of sufficient concentration to be of commercial value are (1) as veins, (2) as replacements of limestone, and (3) as a residual mineral in clays resulting from weathering.

Vein occurrences of the mineral may include (1) barite as a common gangue mineral in veins of metallic ores; or (2) barite as the chief mineral in veins, with little or essentially no development of metallic ores, in sedimentary rocks (sandstones and limestones), in metamorphic crystalline rocks (schists and gneisses), and in igneous rocks. Well-known examples of each of these occurrences of the mineral in the southern Appalachian States are described in another part of this paper. Likewise, the occurrence of barite as a replacement of limestone finds ample illustration in Virginia, where it forms an important type of deposit.

Finally, an important mode of occurrence of barite in the known commercial areas of the United States is as a residual mineral, forming masses of varying sizes and shapes in the clays derived by weathering from the decay of limestones. A large part of the barite mined in the Appalachian States and in Missouri is taken from the residual mode of occurrence of the mineral.

CHARACTER OF ASSOCIATED ROCKS

From the mode of occurrence of barite discussed above, the mineral shows wide range as to kind of rock in which it is found. The numerous occurrences of barite in the Appalachian States afford ample illustration of the wide range in the kind of rock with which it is associated. These include rocks of each of the three major groups—sedimentary, igneous, and metamorphic—and workable concentrations of the mineral are known in each of the three rock groups. Of these, limestones and their residual clays among the sedimentary rocks are vastly the most important sources of supply of the mineral.

Of the sedimentary rocks, limestones and sandstones or their residual clays, especially limestone, are the usual ones which yield workable de-

² Lindgren, W.: *Mineral Deposits*, pp. 98 to 102 (1913). References to the literature cited.

Clarke, F. W.: *Bulletin No. 491, U. S. Geological Survey*, pp. 553 to 555 (1911). References cited.

³ Emmons, W. H.: *Economic Geology*, vol. iii, No. 7, p. 618 (Oct.-Nov., 1908).

posits of the mineral. In at least one locality, however, Prince William County, Virginia, barite is associated with the Triassic red shales and impure limestones as veins. Occurrences of barite in limestone or its residual clays are noted in each of the larger physiographic divisions of the Appalachian States.

The known deposits of barite in igneous and metamorphic siliceous crystalline rocks in the Appalachian region are limited to the Piedmont Plateau, where the mineral occurs as veins, chiefly in schists and crystalline limestones, and occasionally in granite. Examples are known in Virginia, North Carolina, and South Carolina.

ASSOCIATED MINERALS

These will vary for the individual deposit, but the more commonly occurring ones are found to be dependent in large measure upon the mode of occurrence of the barite. In the vein and replacement types of deposits⁴ barite is often associated with metallic sulphides, especially galena, sometimes sphalerite, chalcopyrite, and pyrite; and several of these may be developed in the same deposit. When in quantity, galena is harmful, since it renders the ground product too dark. Quartz, fluorite, and calcite are frequent accompaniments of barite in the deposits of some districts. Fluorite is especially noted in the Kentucky and Tennessee vein occurrences of barite, and in some of the Valley occurrences in Virginia.

In the residual occurrences of barite, the associated minerals will naturally vary with the character of the original rock, but probably the most frequently occurring ones, exclusive of the clay through which the nodular barite is scattered, are the oxides of iron and manganese, and silica in the form of both quartz and chert. In addition to these, tremolite and biotite are noted in some of the Virginia occurrences of barite in the crystalline rocks.

OCCURRENCE AND INDUSTRY BY STATES

In the following pages the barite industry in the Appalachian States is summarized by States. The treatment is by alphabetical arrangement of the States regardless of their rank as producers.

Alabama

The known barite deposits in Alabama are confined to the following five counties of the Paleozoic area in the central and northeastern parts

⁴ No account is here taken of the veins of metallic ores in which barite occurs as a gangue mineral, nor of the occurrences of ore deposits in which barite is developed as a subordinate mineral.

of the State: Bibb, Calhoun, Etowah, St. Clair, and Shelby. The most important localities include (1) Maguire Shoals on Little Cahaba River, at the "Sinks" on Six Mile Creek, and near Pratt's Ferry in Bibb County; (2) near Tampa in Calhoun County; and (3) near Greensport in St. Clair County. Barite also occurs at other places in these counties, and is reported from two localities in Shelby County, but these appear to be of doubtful commercial value.⁵

The Alabama deposits of barite are residual in type, probably concentrated in the clay from veins and replacements in the Cambrian and Ordovician limestones, as the early literature contains references to vein occurrences of the mineral. The mineral occurs in loose pieces on the surface, and in lumps, nodules, and irregular masses imbedded in the residual clays derived from the Knox dolomite of Cambrian and Ordovician age, and the Pelham or Trenton (Chickamauga) limestone of Ordovician age. The deposits seem to be localized along lines of folding and faulting, the more important ones being found near the contact of the Knox dolomite with the Pelham or Trenton limestone, although some of the smaller deposits occur entirely within the limits of the residual clays of the Knox dolomite.

In some of the earlier reports of the State Survey, Dr. Eugene A. Smith, State Geologist, refers to many occurrences of barite in veins. Concerning the occurrence of barite in Bibb County, Smith says:⁶ "Barite, or heavy spar, is of frequent occurrence in the Quebec [Knox] dolomite. Maguire's Shoal on the Little Cahaba, and the 'Sinks' on Six Mile Creek, may be mentioned as localities, but barite is found in veins, in many other places." Again, the same author says of barite in Shelby County:⁷ "Just south of Calera, and associated with beds of limestone of this group [Chazy and Trenton], is a small bed of limonite exposed in the dry channel of a little branch. The limonite has very intimately mixed with it, in varying proportions, *barite* or *heavy spar*. Some portions of the bed showed almost pure limonite, others limonite and barite in equal proportions, and still others, almost pure barite. Scarcely a hand specimen could be obtained without both minerals.

"Barite in veins constantly accompanies the belt of Chazy limestone described above."

In Berney's *Handbook of Alabama*,⁸ Smith states that barite is found in veins in many localities in the Knox dolomite.

⁵ Smith, E. A., and McCalley, H.: *Index to the Mineral Resources of Alabama, Geological Survey of Alabama*, pp. 62, 63 (1904).

Grasty, J. S.: Barite Deposits of Alabama, *The Tradesman*, pp. 35, 36 (July 17, 1913).

⁶ Smith, E. A.: *Report of Progress for 1875, Geological Survey of Alabama*, p. 95 (1876).

⁷ *Idem*, p. 118.

⁸ Berney, Saffold: *Handbook of Alabama: A Complete Index to the State; with a geological map, etc.*, p. 156 (Mobile, Ala., 1878).

The mineral is generally of good quality, usually very pure and white, sometimes grayish and bluish in color, and stained with iron oxide. In some cases the barite is associated with or occurs near limonite deposits, and at times carries more or less galena. Notwithstanding the good quality of the barite in most of the important localities, developments have thus far been slight, and so far as known little or none of the mineral has been placed upon the market. There has been no production of barite in Alabama since 1906. Probably the chief cause for the lack of development of some of the better deposits of the mineral is to be attributed to the absence of a plant in the State to consume the output. The distance and hence freight rates to barite mills in adjoining States are prohibitive for the Alabama product because of a nearer and cheaper source of supply of the mineral.

Georgia

Barite has been mined at Elton, Murray County, and in the Cartersville district, Bartow County, in northwest Georgia. For some years,



FIG. 2.—BARITE MINE NEAR CARTERSVILLE, BARTOW COUNTY, GEORGIA.
(S. W. MCCALLIE, PHOTO.)

however, the entire production has come from the Cartersville district where iron ore, ocher, and barite are mined. (Fig. 2.)

The Elton deposit in Murray County has not been worked in recent years. The composition⁹ of the barite from this locality is: BaSO_4 ,

⁹ *The Mineral Industry*, vol. xix, p. 69 (1910).¹

98.82; Al_2O_3 , 0.2; Fe_2O_3 , 0.33; SiO_2 , 0.27; moisture, 0.38 per cent. The mineral is reported to be lower in iron than that mined in the Cartersville district.

The Cartersville district, occupying the southeastern half of Bartow County, includes about 70 square miles, nearly equally divided between the Paleozoic formations (quartzite, limestone, shale, and dolomite) on the west and the older crystalline and metamorphic rocks (schists, gneisses, slates, and conglomerates) of the Piedmont Plain and Appalachian Mountains on the east.¹⁰

The barite deposits are associated with certain of the Paleozoic formations which occupy the west half of the district, and which named in descending order are: (1) Knox dolomite, (2) Conasauga and Rome formations, (3) Beaver limestone, and (4) Weisner quartzite.

Except the Knox dolomite, these formations belong to the middle or lower Cambrian, and it is probable that the lower portion of the Knox should be classed with the Cambrian. The barite deposits are associated chiefly with the Weisner quartzite and Beaver limestone, though some have been found with the Knox dolomite.

The Weisner quartzite, the basal member of the Cambrian, forms a nearly continuous belt about 19 miles long with an average width of from 1 to 2 miles. It has an estimated thickness of 2,000 to 3,000 ft., and consists of fine-grained quartzite, some beds of fine conglomerate, and interbedded siliceous shale. The conglomerate beds frequently show angular fragments of feldspar, indicating that a part of the material composing it was derived from the granite area to the east. The formation has been intensely folded, and is probably cut by numerous faults, as

¹⁰ For a description of the geology and ore deposits of the Cartersville district, see: Spencer, J. W.: The Paleozoic Group, *Geological Survey of Georgia*, pp. 99 to 107 (1893).

Hayes, C. W.: Geological Relations of the Iron-Ore Deposits in the Cartersville District, Georgia, *Trans.*, xxx, 403 to 419 (1900).

Hayes, C. W., and Phalen, W. C.: A Commercial Occurrence of Barite near Cartersville, Ga., *Bulletin No. 340, U. S. Geological Survey*, pp. 458 to 462 (1908).

McCallie, S. W.: A Preliminary Report on a Part of the Iron Ores of Georgia, *Bulletin No. 10-A, Geological Survey of Georgia*, pp. 109 to 176 (1900).

Watson, Thomas L.: Geological Relations of the Manganese-Ore-Deposits of Georgia, *Trans.*, xxxiv, 207 to 253 (1903).

——— A Preliminary Report on the Manganese Deposits of Georgia, *Bulletin No. 14, Geological Survey of Georgia*, pp. 32 to 99 (1908).

——— The Yellow Ocher Deposits of the Cartersville District, Bartow County, Georgia, *Trans.*, xxxiv, 643 to 666 (1903).

——— A Preliminary Report on the Ocher Deposits of Georgia, *Bulletin No. 13, Geological Survey of Georgia* (1906). 81 pp.

Maynard, T. P.: A Report on the Limestones and Cement Materials of North Georgia, *Bulletin No. 27, Geological Survey of Georgia*, (1912). 293 pp.

Watson, T. L., and Grasty, J. S.: The Geology and Barite Deposits of the Cartersville, Georgia, District, *The Tradesman*, May 22, 1913, pp. 35 to 37.

evidenced by its crushed and brecciated condition in many places. So extensive is the crushing and brecciation in some of the larger mine openings in the district that the original bedding of the quartzite is entirely obscured. The resulting structural conditions of the quartzite have been particularly favorable to chemical action and are responsible for the deposition of the mineral deposits, especially ocher and barite, contained in it.

A chemical analysis made by the N. P. Pratt laboratory on specimens collected by the senior writer from different exposures of the formation gave 4.46 per cent. of barium sulphate.

Stratigraphically above, but topographically below the Weisner quartzite, is a thickness of 800 to 1,200 ft. of a gray crystalline magnesian limestone, which is shaly in places and at times chert-bearing, known as the Beaver limestone. This limestone forms a narrow belt of lowland along the western margin of and parallel to the ridges of the harder Weisner quartzite. Outcrops of the limestone are seldom seen and its surface is covered by a deep mantle of residual red clay by which the formation may be readily traced.

Barite has two modes of occurrence in the Cartersville district: (1) As deposits from solution in fractures and cavities in the Weisner quartzite in intimate association with the deposits of yellow ocher, and (2) as crystalline nodules and masses of varying size imbedded in the residual clays derived from the decay of the Weisner quartzite and Beaver limestone. Each of these has proved to be a source of supply of barite, but the principal commercial source of the mineral has been from the residual clays. In addition to yellow ocher, the barite deposits of the Cartersville district are more or less closely associated with iron ore.

Barite is found in nearly all the ocher deposits of the Cartersville district. It is known to the miners as the "flowers of ocher," and since it remains in the residual soils derived from the quartzite, it affords the best means of tracing the ocher deposits. It has been shown by Hayes, and later confirmed by Watson, that the ocher deposits with which barite is associated are the direct replacements of the silica in the Weisner quartzite from thermal solutions circulating along the zones of fracture. These relations are best indicated at the mine of the Georgia Peruvian Ocher Co., at the wooden bridge over Etowah River. Here large clusters and groups of magnificent barite crystals are found in the pockets of ocher and in the fractures and cavities of the quartzite. The usual occurrence of barite is in divergent groups of massive tabular crystals, giving a crested appearance and grading into both straight and curved laminated masses. Groups of crystals weighing 25 lb. and more are not uncommon. It seems probable, as Hayes states, that the crystals of barite "were probably deposited after the conditions favorable for the solution of silica and the deposition of ocher had passed. Groups of acicular

crystals of this mineral, several inches in length, are not uncommon. It also occurs in white granular veins."¹¹

A second and commercially more important occurrence of barite in the Cartersville district is as nodules and masses of varying sizes and shapes imbedded in the residual clays. Deposits of this type of the mineral are rather widely distributed over the district, but probably many of them will not prove profitable because of lack of sufficient concentration for mining alone. However, a number of promising deposits have been opened, but further development is necessary before a definite statement can be made as to the extent, quality, and workability of the deposits in the district.

The principal barite deposits occur near the south end of the quartzite belt where the structure has been shown to be anticlinal. The deposits are found on the east side of the anticline lower down the slope but adja-

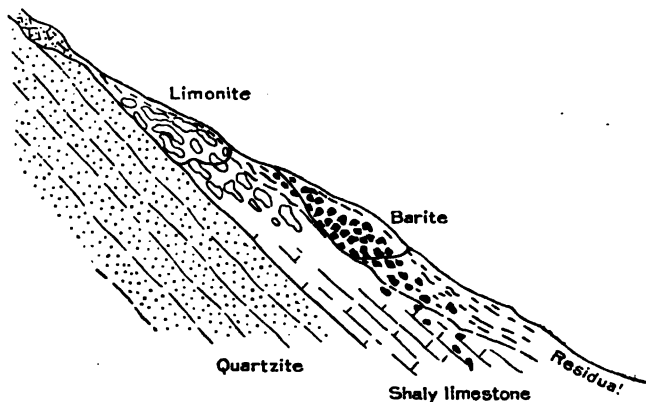


FIG. 3.—SKETCH SECTION SHOWING RELATIONS OF BARITE AND LIMONITE TO UNDERLYING FORMATION NEAR CARTERSVILLE, GA. (AFTER HAYES AND PHALEN.)

cent to a belt of limonite which rests directly on the quartzite and has been mined for many years. According to Hayes and Phalen,¹² "while this belt has not been sufficiently prospected to prove the continuity of the deposits, their thickness is indicated by a tunnel which has been driven into the side of the hill about 300 ft. north of the present workings. This tunnel penetrates 120 ft. of the barite-bearing clay and then enters the iron ore." (See Fig. 3.)

Concerning the relations of the limonite and barite to the underlying formations Hayes and Phalen say:¹³ "One striking feature is the sharp separation between the two minerals; each appears to occupy a definite horizon, and so far as observed there is no intermingling. While both in

¹¹ Hayes, C. W.: *Trans.*, xxx, 418 (1900).

¹² Hayes, C. W., and Phalen, W. C.: *Bulletin No. 340, U. S. Geological Survey*, p. 461 (1908).

¹³ *Idem*, p. 461.

their present condition are residual, they appear originally to have replaced distinct beds in the shaly limestone overlying the quartzite, the iron being deposited immediately adjacent to the quartzite, and the barite, for the most part, at least, in higher beds. It is highly probable that deposition of the barite and limonite took place, in part, at least, simultaneously. It is quite certain that gravity has been largely instrumental in concentrating the former into a workable deposit."

The barite deposit is described as having a thickness of 50 ft. normal to the surface slope; is intermingled with residual clays containing fragments of quartzite; and the barite makes up about a third of the material mined. The barite forms nodules weighing several ounces up to boulders of several hundred pounds; is mostly massive and compact granular in texture; and of pure white to bluish color.

*Kentucky*¹⁴

Barite is reported to have been first marketed from Kentucky in 1903, when shipments were made from western Kentucky and from Scott County in central Kentucky. With the exception of a small tonnage in 1907 from western Kentucky, the production has been from the central part of the State.

In Kentucky barite deposits are known in (1) the central and (2) the western parts of the State. Thus far mining of barite in the State has been confined chiefly to the central or Blue Grass region, which comprises the following 16 counties: Anderson, Bourbon, Boyle, Clark, Fayette, Franklin, Garrard, Harrison, Henry, Jessamine, Lincoln, Madison, Mercer, Owen, Scott, and Woodford. Of these, Boyle, Fayette, and Garrard counties, in the order named, are the principal producers of barite. The map, Fig. 4, shows the location of the principal barite deposits in the central Kentucky region.

The barite deposits of central Kentucky are confined to the Ordovician and are most prominently developed in the lower or Mohawkian division of the Ordovician, more especially in the limestones of Stones River and Trenton age and in the Eden shale. The rocks are all sedimentary and

¹⁴ Fohs, F. Julius: Barytes Deposits of Kentucky, *Kentucky Geological Survey*, ser. iv, vol. i, pt. 1, pp. 441 to 588 (1913).

Grasty, J. S.: Barite Deposits of Kentucky, *The Tradesman*, July 3, 1913, pp. 33, 34.

Miller, A. M.: The Lead and Zinc Bearing Rocks of Central Kentucky, with Notes on the Mineral Veins, *Bulletin No. 2, Kentucky Geological Survey* (1905), 35 pp.

Ulrich, E. O., and Smith, W. S. T.: Lead, Zinc, and Fluorspar Deposits of Western Kentucky, *Bulletin No. 213, U. S. Geological Survey*, pp. 205 to 213 (1903).

— The Lead, Zinc, and Fluorspar Deposits of Western Kentucky, *Professional Paper No. 36, U. S. Geological Survey* (1905), 218 pp.

The Barite Deposits of Western Kentucky and Union County will be described in *Bulletin No. 15*, to be issued shortly by the State Geological Survey.

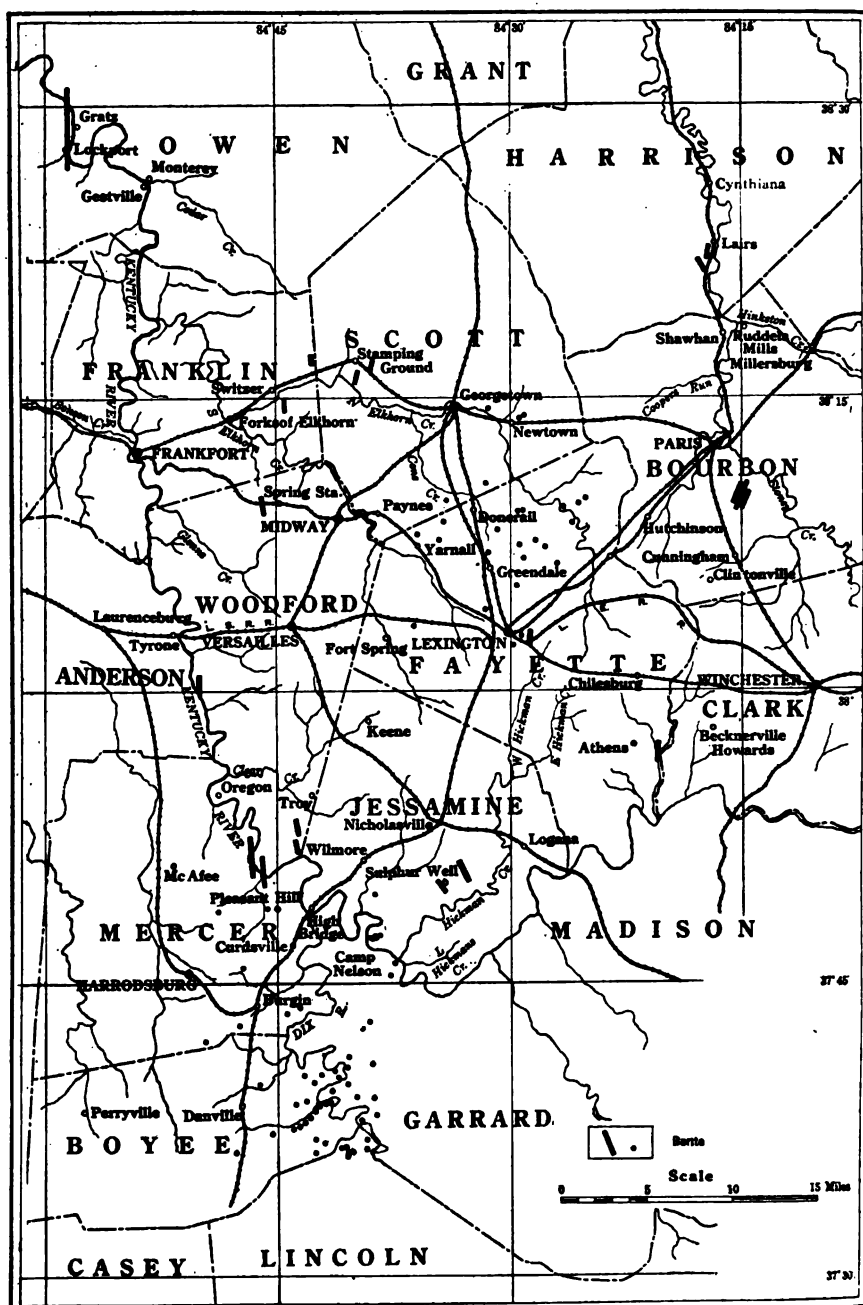


FIG. 4.—PRINCIPAL BARITE DEPOSITS IN CENTRAL KENTUCKY. BARITE SHOWN BY SOLID BLACK DOTS AND DASHES. (ADAPTED FROM PUBLISHED REPORTS OF KENTUCKY GEOLOGICAL SURVEY.)

the important deposits of barite are confined to the limestone members. The barite occurs chiefly in veins filling simple fissures or faults, chiefly the latter. The veins are vertical or slightly inclined. They vary in width up to 24 ft., but the usual width is from 1 to 3 ft. The thickness of individual veins varies much from point to point, swelling or pinching both laterally and vertically. They are apt to become lean and barren in the argillaceous beds, but are of good grade in the limestone beds. According to Miller, the veins seem to have considerable vertical extent, and mining operations on some of them have extended to depths ranging from 100 to 325 ft., without noticeable tendency to close up.¹⁵

The barite deposits of central Kentucky are either of the fissure, breccia, or replacement type, or a combination of these. The breccia deposits are associated with faults. The principal minerals associated with the barite are calcite, fluorspar, sphalerite, and galena, with quartz, marcasite, pyrite, chalcopryrite, smithsonite, limonite, and manganese oxide less frequent. Strontianite and celestite are reported to occur (see analyses below).

The following analyses¹⁶ of barite from the central Kentucky region are of interest:

Analyses of Kentucky Barite

	I	II	III	IV	V	VI	VII	VIII	IX	X	XI
Ignition.....	0.17	0.34	0.87	0.28	0.62	0.23	1.40	0.65	0.15	1.19	0.28
SiO ₂	0.34	0.11	0.11	0.25	0.10	0.05	0.55	0.18	0.10	0.20	0.23
FeO ₂	0.06	0.02	0.04	0.06	0.06	0.03	0.06	Trace	0.04	0.08	Trace
BaO.....	50.86	50.96	60.26	64.72	59.80	50.96	54.95	10.71	64.68	49.98	60.55
SrO.....	8.99	11.65	3.80	Trace	2.41	11.10	6.75	45.06	0.20	11.07	3.72
CaO.....	3.61	Trace	0.76	Trace	2.09	0.88	0.20	0.25	Trace	Trace	Trace
SO ₂	33.87	36.31	34.60	33.82	33.23	35.66	34.22	42.28	33.97	35.12	35.38
F.....	2.45	Trace	0.52	0.00	1.42	0.60	0.14	0.00	0.00	0.50	0.00
ZnO.....	Trace	0.00	0.00	0.00	0.00	0.00	1.23	0.00	0.00	1.50	0.00
Total.....	100.25	99.39	100.96	99.13	99.73	99.51	99.50	99.13	99.14	99.68	100.16

I. John Baughman prospect, one-half mile north of Danville, Boyle County.

II. Lee River prospect, one-half mile south of Harrodsburg, Mercer County.

III. Farris prospect, 4 miles northeast of Danville, Boyle County.

IV. Meyers prospect, near Mexico, Crittenden County.

V. Ray mine, 2 miles from Fredonia, Caldwell County.

VI. Ray mine, 2 miles from Fredonia, Caldwell County.

VII. Hayden open cut, 1½ miles south of Danville, Boyle County.

VIII. Celestite from Truesdale, Owen County.

IX. Perkins prospect, 1½ miles southeast of Burgin, Mercer County.

X. Mosby prospect, 1½ miles south of Shryock Ferry, Woodford County.

XI. Shelton open cut, 4 miles east of Danville, Boyle County.

¹⁵ According to Fohs, the veins "may reach a depth of over 1,200 ft., a length of four or five miles and a width of 16 ft. The average present depth of the known deposits is 600 ft., at least half of which carries a commercial product."

¹⁶ Fohs, F. Julius: *Kentucky Geological Survey*, ser. iv, vol. i, pt. 1, pp. 449 to 451 (1913).

Attention is especially directed to the large but variable amount of strontium oxide reported in most of the analyses.

In the western Kentucky region barite is found in the following eight counties: Caldwell, Christian, Crittenden, Cumberland, Livingston, Lyon, Russell, and Trigg, but the most important deposits are confined to three of these, namely, Caldwell, Crittenden, and Livingston. This area forms the well-known fluorspar district of western Kentucky, and thus far the barite deposits are incidental to those of fluorspar, just as fluorspar is incidental to the barite deposits of central Kentucky. Most of the deposits have been worked for fluorite and the sulphides, since barite is not particularly abundant. According to Fohs, the most important deposits of barite in the western Kentucky region are: "Commercial shaft, Meyers, Sullenger, and Bibb in Crittenden County; the Lindley, Stone, and Bradshaw in Livingston County; and the Satterfield, Ray, and Lowery in Caldwell County."

The barite of western Kentucky is found chiefly as a vein mineral in the fluorspar deposits, which, according to Smith, occur as (1) vein deposits filling fissures due to faulting; (2) ores cementing breccias; and (3) metasomatic replacements. The vein deposits (fluorite, barite, some galena, and some sphalerite) filling faults are by far the most important occurrence. The faults strike northeast, northwest, and east, though a few unimportant ones have a northerly direction. The veins occur in the formations of the Chester and Meramec groups, especially in Ste. Genevieve and St. Louis limestone, of Carboniferous (Mississippian) age. In Crittenden County vertical dikes and horizontal sheets of peridotite occur ranging up to 25 ft. in width. The veins dip at a high angle, usually 60° or more, and are subject to change both in dip and strike. They show considerable variations in width, but the most important ones do not exceed 6 or 8 ft., and many of them are reported to pinch out completely for short distances. Besides angular fragments of the country rock, the veins are composed of the minerals fluorite, calcite, barite, sphalerite, and galena, occasionally quartz and pyrite, and rarely others. Banding is a pronounced structural feature.

Concerning the occurrence of barite, Smith says:¹⁷

"Barite is common in some of the veins, although in many if not most of them it is not found at all. Taking the district as a whole, however, it is probably next in abundance to calcite as a vein mineral. In many cases where it occurs it forms only a small proportion of the vein, although occasionally it is the chief constituent, as at the Lowery mine, at one of the Myers prospects, and at the Bateman prospect. Seams in brecciated rocks and narrow veins sometimes consist wholly of this mineral.

"Barite occurs in all of the rocks in which fluorite is found—in Chester quartzite and in both Ste. Genevieve and St. Louis limestone. Where seen only in small amounts it was, as a rule, along the margin of the vein and never distributed through it. When occurring with fluorite in seams, it sometimes shows symmetrical banding.

¹⁷ *Professional Paper No. 36, U. S. Geological Survey*, p. 135 (1905).

"The only vein of this mineral which has been developed to any extent is that on which the Ray and Lowery shafts are situated. Here the vein has a width of several feet near the surface, pinching to from 6 inches to a foot at a depth of 30 or 35 feet. It consists almost entirely of barite, with a very small proportion of fluorite and calcite. It is more or less banded, and locally one of the central bands, from a fraction of an inch to 2 inches in width, contains a considerable amount of medium-grained sphalerite."

The various writers who have studied the western Kentucky vein deposits are generally agreed that they were formed from solution, and that the source of the minerals has been from the limestones of the region. It is not impossible that the source of some of the minerals at least may have been the magma from which the peridotite dikes were derived. Smith says:¹⁸ "It is not probable, however, that the dikes themselves are the source of the fluorite, chiefly on account of the total volume of the fluorite veins as compared with that of the igneous dikes and the very small fluorine content of the peridotite."

Maryland

In Maryland barite occurs in Frederick and Carroll counties, and small occurrences have been reported in the Hagerstown Valley which are sim-

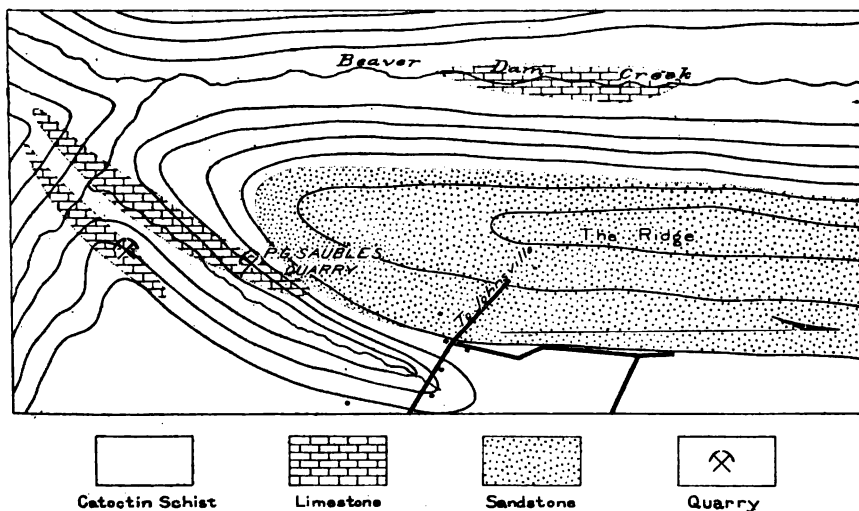


FIG. 5.—MAP OF SAUBLE QUARRY AND VICINITY, NEAR UNION BRIDGE, MARYLAND.

ilar to those found to the northeast in Pennsylvania. However, the deposits in Frederick and Carroll counties give the greater promise of eventually becoming of economic importance. The deposit here de-

¹⁸ *Idem*, p. 151.

scribed may, broadly speaking, be considered as typical. Its association with the Piedmont limestones suggests that further search for barite in this particular physiographic province (Piedmont) would be worth while. Deposits similar to that on the Sauble property occur not only in the Piedmont limestones of Virginia but are also found at the same geologic horizon in North Carolina and South Carolina.

The Sauble property is located on a small feeder of Beaver Dam Creek in Frederick County and to the east of the large quartzite ridge extending in a northeast-and-southwest direction between Johnsville and Union Bridge. Structurally, the rocks of this ridge appear to be compressed into a syncline overturned to the westward. The quartzite is tentatively

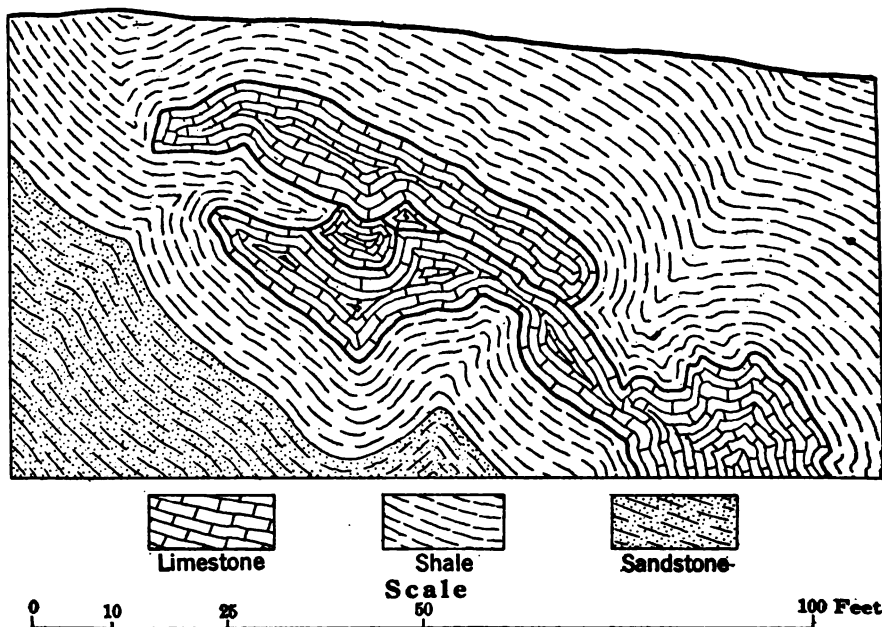


FIG. 6.—SECTION AT SAUBLE QUARRY, MARYLAND.

correlated with the Weverton formation and is underlain by the Loudon, which is composed here of limestone and shale. (Figs. 5 and 6.)

The barite on the Sauble property occurs in the shape of small bands, stringers, etc., in the Loudon limestone, which, as would be inferred from the structural relations referred to above, is greatly fractured and crushed, and in the overlying shale fracturing and crushing are even more emphasized. The quarry opened here has never been worked, however, for barite, but was operated with a view to obtaining limestone for calcination for local use, with no attention given to the barite. Judging from the lumps and small piles of barite lying about, it would seem that, to some extent at least, the mineral was sorted out and thrown aside.

Barite is also found at other localities in the Piedmont province of Maryland, and is characterized by occurring in crushed zones and along lines of dislocation which resulted from movements that profoundly compressed and more or less altered all the rocks of the region. These limestones were formerly much more widely distributed, but, in general, only the portions which have been compressed and preserved by reason of their synclinal structure now remain, and in most instances they have been greatly crushed and fissured, and recemented by calcite, and here and there by barite.

*North Carolina*¹⁹

North Carolina is one of the important barite-producing States in the Appalachian region. Deposits of barite occur in Gaston and Orange counties in the Piedmont region, and in Madison County in the western part or mountainous region of the State, but the production is confined to Gaston and Madison counties.

In Orange County, barite of good quality is found about 3.5 miles southeast of Hillsboro, near the Chapel Hill road. Some development work was carried on in 1901.

Barite is found at several localities in the vicinity of King's and Crowder's mountains, but the principal producing mine is located about 5 miles south of Bessemer City, Gaston County, near the north end of Crowder's Mountain. The barite occurs in quartz-sericite schist as veins ranging in thickness from 2½ to 6 ft. as exposed in the workings. The veins strike about N. 10° to 15° E., and dip at high angles, usually toward the west; in some places they are nearly vertical. Mining of barite has been carried on at a number of places in the vicinity of the Lawton mine, as shown in many old workings. Cyanite schist outcrops near the veins on the west side and lenses of magnesian limestone occur in the region, but so far as observed are not found in association with the barite.

The barite is granular, sometimes coarsely crystalline, and in places cleavage surfaces 2 in. across are noted. The associated minerals are quartz, galena, sphalerite, pyromorphite, and stains of iron and manganese oxides. According to Pratt, the veins are lenticular in shape, and probably represent the filling of fissures and crevices in the schists, which may have been caused by the faulting or tearing apart of the rock.

In Madison County in western North Carolina, the barite deposits occupy an area about 5 miles long extending a little north of east from Bluff on Spring Creek to and across French Broad River. Within this area barite has been worked chiefly in the vicinity of Marshall, Stack-

¹⁹ Keith, Arthur: *Asheville Folio, North Carolina-Tennessee*, No. 116, U. S. Geological Survey, p. 9 (1904).

Pratt, J. H.: *Economic Paper No. 6, North Carolina Geological Survey*, pp. 62 to 65 (1902).

house, Sandy Bottom, and Hot Springs. According to Keith, the barite occurs as irregular veins in pre-Cambrian granites and to a less extent in

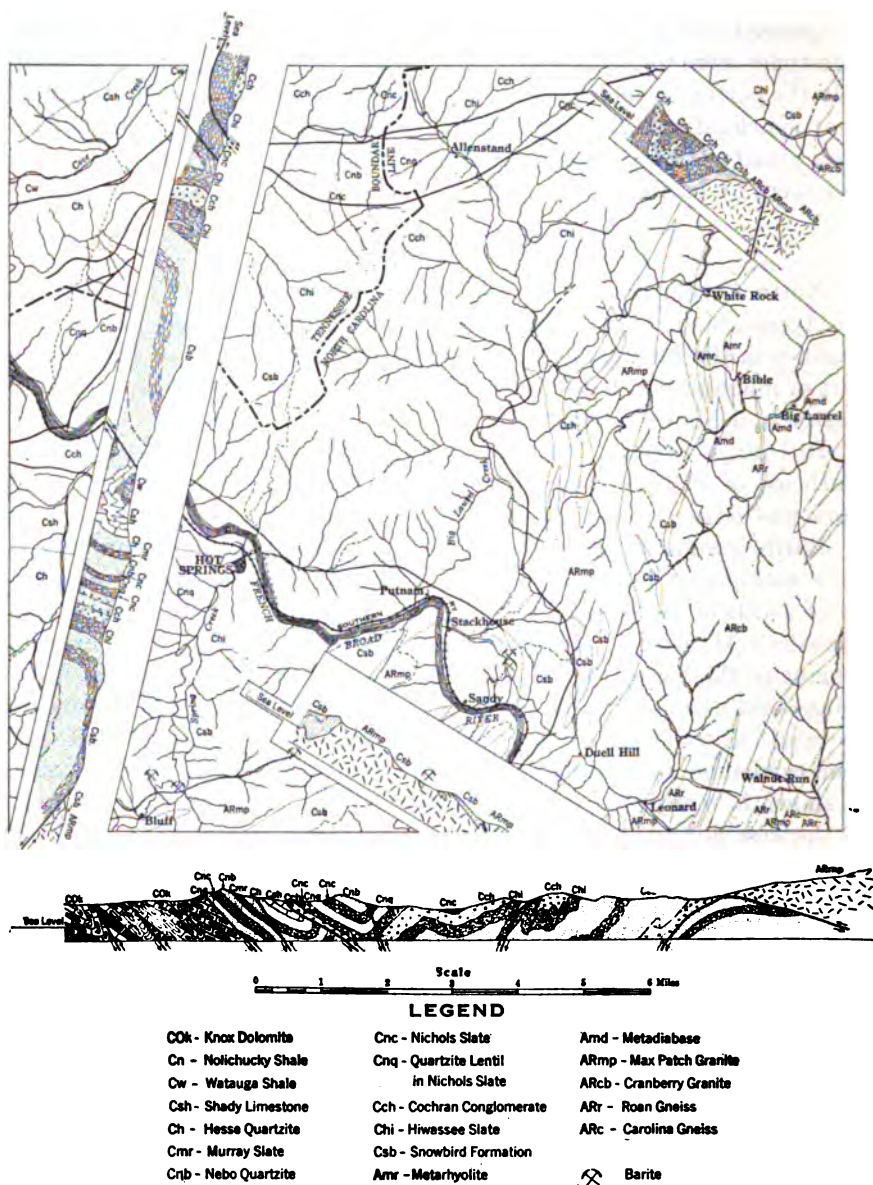


FIG. 7.—MAP AND SECTIONS OF THE BARITE AREA IN WESTERN NORTH CAROLINA. (*Asheville Folio North Carolina-Tennessee, No. 116, U. S. Geological Survey.*)

Cambrian feldspathic quartzite. Watkins²⁰ reports that some of the deposits are associated with limestone. Pratt states that the veins vary

²⁰ Personal communication, October, 1914.

from 3 to 10 ft. in thickness and the barite is practically free from other minerals. At the Stackhouse mine Pratt reports that outcroppings of barite have been observed on the surface for a distance of 2,200 ft. The map, Fig. 7, gives a good general idea of the geology of the barite area in western North Carolina.

Keith gives the following description of the deposits:²¹

"The country rock was much broken and shattered before the formation of the barite, and the spaces were filled with the ore. From the arrangement and crystalline condition of the barite, it is evident that it was deposited from aqueous solutions. As to the derivation of the material in solution there is no evidence; probably it came from the great mass of granite within which it is now found. All of these barite areas occur in a belt near and parallel to one of the principal thrust faults of the region. The rocks were greatly disturbed during the production of the original fault plane, and later, to a less extent, during its secondary deformation. It appears most likely that the crushing of the quartzites seen in the barite deposits was associated with this later deformation. In the old workings on Spring Creek, near the level of the creek, the barite-bearing quartzites are very close to the thrust fault and there seems to be a distinct connection between the fault and the barite. The barite occurs in large crystals and crystalline masses which have not been deformed, as were the minerals of the inclosing rocks. From this it is clear that the barite was deposited after the period of faulting and folding, and although the shattering of the rock may have been due to the folding, the barite must have been introduced later. The crevices in the shattered granite afforded comparatively easy access to the solutions that carried the barite, but no direct connection in origin can be traced between the faults and the barite deposits.

"At present only the deposits on Spring Creek are being mined. A considerable amount of barite was taken from the veins a mile southeast of Stackhouse, but the deposits on Doe Branch were never developed to any extent. Southeast of Stackhouse small open cuts were made and short shafts and tunnels were driven. These were nearly on the crest of a ridge, and penetrated for some distance the more or less weathered granites. On Spring Creek the mining has been done chiefly through open cuts. These are now of considerable size and much material has been removed. The openings lie on ridges which are about 500 feet above water level and which slope steeply toward Spring Creek. The surroundings are thus favorable for extensive operations, and the quantity of barite appears to be large."

*Pennsylvania*²²

Barite is not mined on a commercial scale in Pennsylvania at present, although small shipments have been made from the Cumberland Valley in the southern part of the State in the vicinity of Waynesboro, Franklin County. Barite is also reported from west of New Hope, Butler County; at Phoenixville mines, and in small quantity in old Jug Hollow mine, Schuylkill township, Chester County; formerly mined at Fort Littleton, Fulton County; and at the Perkiomen copper mines, Montgomery County.

²¹ Keith, Arthur: *Asheville Folio, North Carolina-Tennessee*, No. 116, U. S. Geological Survey, p. 9 (1904).

²² Stose, G. W.: *Bulletin No. 225, U. S. Geological Survey*, pp. 515, 516 (1904).

Sanford, S., and Stone, R. W.: *Useful Minerals of the United States, Bulletin No. 585, U. S. Geological Survey*, p. 158 (1914).



FIG. 8.—BARITE MILL OF THE CAROLINE BARYTES CO., STACKHOUSE, N. C.
(J. H. WATKINS, PHOTO.)



FIG. 9.—BARITE MILL OF THE CHEROKEE CHEMICAL CO., KING'S CREEK, S. C.
(J. H. WATKINS, PHOTO.)

In the vicinity of Waynesboro the barite is associated with Cambrian limestones and their residual clays. Openings have been made to the north, east, and northeast of Waynesboro both in the residual clays and in the limestone. The deposits are found in the folded and brecciated portions of the limestone, and the barite occurs in the brecciated rock as a vein filling, but such deposits are not considered profitable. Stose says of the barite in the two openings made in bedrock (limestone) visited by him: "Here the barite is chiefly massive, banded, sub-crystalline or granular, and milky, resembling chert, but is in part clear and crystalline." In the residual deposits the lumps or masses of barite average 6 to 8 in. in diameter, are rough in outline with evidence of dissolved limestone fragments, and are considerably weathered.

*South Carolina*²³

The known commercial deposits of barite in South Carolina are confined to Cherokee County on the North Carolina line. The production has been small and irregular, not, however, on account of lack of quantity and quality of the mineral. The deposits are located in the vicinity of Kings Creek, a station on the Blacksburg-Yorkville branch of the Southern Railway, and are about 13 miles southwest of the Lawton mine in Gaston County, North Carolina (see p. 363).

The barite occurs in quartz-sericite schist as lenticular veins. Thus far limestone has not been observed in association with the barite deposits. The development work done comprises open cuts and pits, which expose in places as many as three parallel veins ranging from 1 to 3 ft. in thickness. Watkins reports the greatest thickness observed was more than 6 ft. of massive white barite exposed in an open cut north of the railroad. The veins have a general strike of about N. 25° E., and dip about 40° to 45° southeast. On the property of the Cherokee Chemical Co., the veins east of the railroad dip to the southeast while those on the west side dip to the northwest, indicating erosion of an anticlinal fold. They conform roughly to the structure of the inclosing schists, and sharp contacts between vein and country rock are usually shown, though small inclusions of the schist are incorporated in the barite in places. Discordance of strike is noted in places, as the veins cut the schists at a slight angle. Branching veinlets are given off from the main veins which fill fractures that cross the structure of the schists. (See Fig. 8.) The field evidence indicates that the veins have had a similar genesis to those farther northeast in Gaston County, North Carolina (see p. 363).

The barite is of good quality, massive granular in structure, and varies

²³ Sloan, Earle: Catalogue of the Mineral Localities of South Carolina, *Bulletin No. 2, ser. iv, South Carolina Geological Survey*, pp. 125, 126 (1908).

Unpublished data furnished by Joel H. Watkins, geologist for the Southern Railway Co., Washington, D. C.

from pure white to pink in color, the white predominating. The pink is mixed with the white barite in milling, since there is apparently no difference in quality of the two differently colored materials. Some parts of the veins are very pure, being singularly free from associated minerals, but other parts contain a little galena and less fine-grained quartz. Sloan states that occasional inclusions of pyrite cause local areas of stain. The barite is generally of good bleaching quality.



FIG. 10.—SMALL VEINLET OF BARITE FROM A 3-FT. VEIN, CUTTING ACROSS STRUCTURE OF SCHIST. CHEROKEE CHEMICAL CO., KINGS CREEK, S. C. (J. H. WATKENS, PHOTO.)

The mill of the Cherokee Chemical Co. is located directly alongside the railroad track and the deposits have been opened at a number of points both to the north and south of the mill, with most of them lying within a radius of half a mile therefrom. The following analysis by Ricketts & Banks of a representative sample of the white barite collected by Banks from the property of the Cherokee Chemical Co., located at Kings Creek, indicates its composition: BaSO_4 , 93.59; Fe_2O_4 , 0.32; CaO , 0.02 per cent.; MgO , trace; SiO_2 , 4.76 per cent.

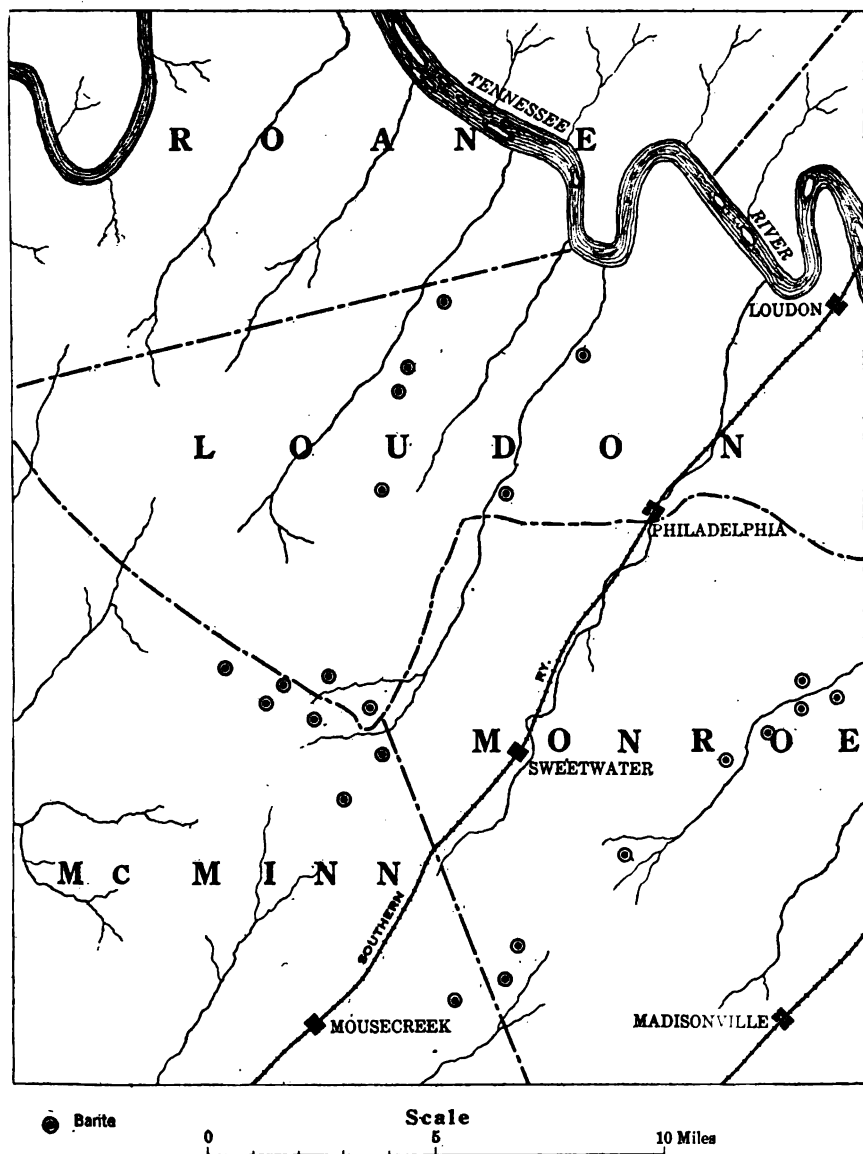


FIG. 11.—MAP OF THE SWEETWATER DISTRICT, TENNESSEE, SHOWING LOCATION OF BARITE WORKINGS. (ADAPTED FROM THE TENNESSEE GEOLOGICAL SURVEY.)

*Tennessee*²⁴

In Tennessee barite is found in many localities, but the more important deposits occur in the eastern part of the State and are grouped into two districts: (1) The French Broad district, which includes parts of Cocke and Sevier counties on the North Carolina line, and (2) the Sweetwater district, centered about the town of Sweetwater and including parts of Loudon, McMinn, and Monroe counties. Barite has been reported



FIG. 12.—1,000 CARLOS OF BARYTES, SWEETWATER, TENN. (G. F. GREENE, PHOTO.)

mined also in Bradley, Jefferson, Smith, and Washington counties; and it occurs as a gangue mineral in a lead mine near Haysborough, Davidson County, and as veins in dolomite 12 miles from Greeneville, Green County.

²⁴ Fay, A. H.: Barytes in Tennessee, *Engineering and Mining Journal*, vol. lxxxvii, No. 2, p. 137 (Jan. 9, 1909).

Grasty, J. S.: The Barite Deposits of Tennessee, *The Tradesman*, vol. lxi, pp. 34 to 38 (1913).

Henegar, H. B.: Barite Deposits in the Sweetwater, Tennessee, District, *The Resources of Tennessee, State Geological Survey*, vol. ii, pp. 424 to 429 (1912).

Hersig, C. S.: Tennessee Barytes, *The Mineral Industry*, vol. x, p. 58 (1901).

Judd, E. K.: The Barytes Industry of the South, *Engineering and Mining Journal*, vol. lxxxiii, No. 16, pp. 751 to 753 (Apr. 20, 1907).

Watson, Thomas L.: Fluorite and Barite in Tennessee, *Trans.*, xxxvii, 890 (1906).

Weller, C. A.: Barytes Mines of the Commercial Mining and Milling Company (Tennessee), *Engineering and Mining Journal*, vol. lxxxiii, No. 18, p. 851 (May 4, 1907).



FIG. 1.—GEOLOGICAL MAP AND SECTIONS OF SWEET-WATER DISTRICT, TENNESSEE.
(London Folio, Tennessee, No. 25, U. S. Geological Survey.)

The French Broad district includes parts of Cocke and Sevier counties, located on the North Carolina line in the mountainous region of the State, south and a little east of Knoxville. The barite occurs in veins from 1 to 6 ft. in thickness, which closely resemble those of Madison County, North Carolina, but owing to lack of transportation the deposits have only been worked to a limited extent. Thus far, the principal developments have been near Wolf Creek in Cocke County. The deposits are reported to contain ample reserves of the mineral, and they carry practically none of the impurities associated with the residual deposits of barite in the Sweetwater district.

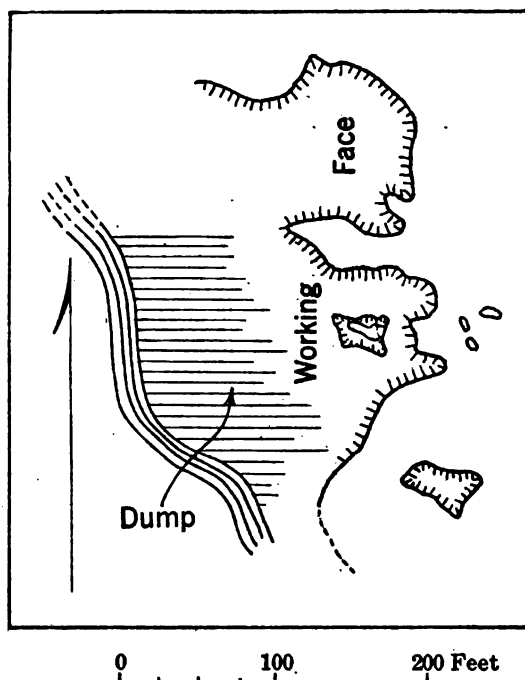


FIG. 14.—SKETCH MAP OF BARITE WORKINGS ON TERRY PROPERTY.

The Sweetwater district is the largest and at present the only important producer of barite in Tennessee. It includes parts of Loudon, McMinn, and Monroe counties, centered within a radius of 20 miles about the town of Sweetwater on the Knoxville division of the Southern Railway, 42 miles southwest of Knoxville. (See Fig. 9.) There has been extensive development and a considerable production of barite in the past. The principal shipping point is Sweetwater, where is located the mill of Gilman & Son for the treatment of the mineral, but other shipping points located on the same line of railway include Philadelphia, Reagan, and Niota. Most of the barite is graded and shipped to the Eastern markets.

(See Fig. 10.) Several companies operate in the district, but the mineral is handled almost entirely by farmers who work between crops.

The barite deposits of the Sweetwater district are of the residual type, occurring as nodules and boulders of varying sizes and shapes in the clays derived from the weathering of folded and faulted rocks which range in age from Cambrian to Ordovician. (See map, Fig. 11.) The principal workable deposits, however, are found in the residual clays derived from the Knox dolomite. In places the mantle of residual clay, in which

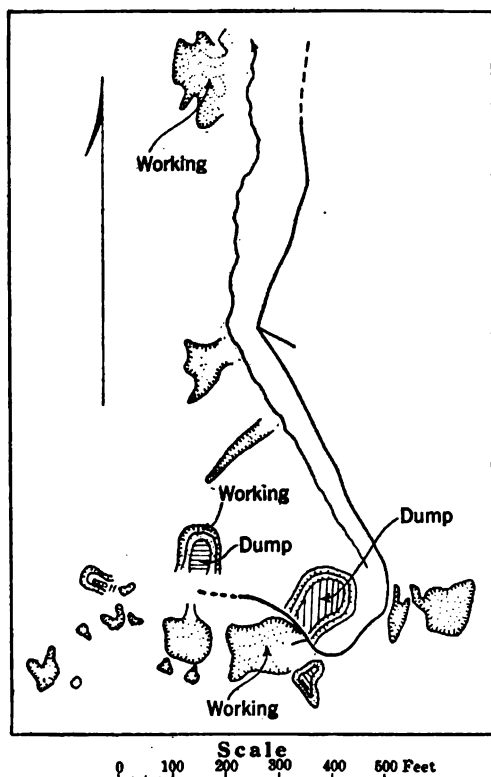


Fig. 15.—METHOD OF MINING BARITE FROM CLAY IN TENNESSEE, RAY PROPERTY.

are assembled the nodules and boulders of barite, is 60 ft. and more in thickness, but usually averages from 10 to 15 ft. Only in one or two workings have the excavations extended down to the hard rock limestone.

A large percentage of the barite mined in the district is coated with limonite, which on account of the slight difference in specific gravity of the two minerals is difficult to remove even by jiggling. Freeing of the barite from the iron oxide before grinding is necessary, as it is very difficult to bleach the mineral thoroughly afterward. Besides iron oxide the barite is intimately associated with chert.

An analysis of a sample of barite taken from the Roy property gave: BaSO_4 , 97.56; Fe_2O_3 , 0.42; SiO_2 , 0.41 per cent. A less favorable sample from the Terry property gave on analysis: BaSO_4 , 94.12; Fe_2O_3 , 0.70; SiO_2 , 2.45 per cent.

Mining of barite in the Sweetwater district is carried on entirely by open-pit work with pick, shovel, wheelbarrow, and wagons. Figs. 12 and 13 serve to show the general character of mining in winning barite from residual clays in the district. In some cases the clay encountered is indurated, and because of its toughness it is necessary to break it down by an explosive. This, however, is the exception, although an explosive might be used at times to advantage in even the less-hardened clays.

Virginia²⁵

In Virginia, barite has been mined since 1845. In the Piedmont region the mineral has been mined in Bedford, Campbell, Louisa, Pittsylvania, and Prince William counties; and in the Valley region in Montgomery, Russell, Smyth, and Tazewell counties. The barite deposits occur in three unlike areas: (1) In the red sandstone-shale series of the Triassic, (2) in the old crystalline metamorphic pre-Cambrian and Cambrian rocks, especially in some of the crystalline limestone areas, and (3) in the Valley region of folded and faulted Cambrian and Ordovician limestones. Areas (1) and (2) compose the Piedmont province, which extends eastward from the Blue Ridge to the fall-line or the western margin of the Coastal Plain sediments. The map, Fig. 14, shows the distribution of worked barite areas in Virginia.

Thus far the Prince William County deposit near Catlett, in the northern part of the State, is the only one in the Triassic rocks that has been worked to any extent. It has been idle since 1903. The barite occurs as a filling or cement of crushed and fractured impure limestones and red shales. The widest of the barite-filled fractures is 4 to 8 ft. The barite is of good white quality, finely to coarsely crystalline massive, and is largely free from most of the common impurities. The mineral was mined by shafts and open cuts, the greatest depth reached being 108 ft.

The principally worked barite deposits of Bedford, Campbell, and Pittsylvania counties, in the south-central part of the State, are centered about Evington, Campbell County, and Toshes and Sandy Level, Pittsylvania County, as shown on map, Fig. 15. The area is one of greatly altered crystalline rocks formed from old sedimentary and igneous rocks.

²⁵ Watson, Thomas L.: *Geology of the Virginia Barite-Deposits*, *Trans.*, xxxviii, 710 to 733 (1907).

—: *The Geology of the Virginia Barite Deposits*, *The Tradesman*, Mar. 20, 1913, pp. 38 to 41, Mar. 27, 1913, pp. 31 to 33.

Campbell, M. R.: *Bristol, Virginia-Tennessee*, *Folio*, No. 59, *U. S. Geological Survey* (1899).

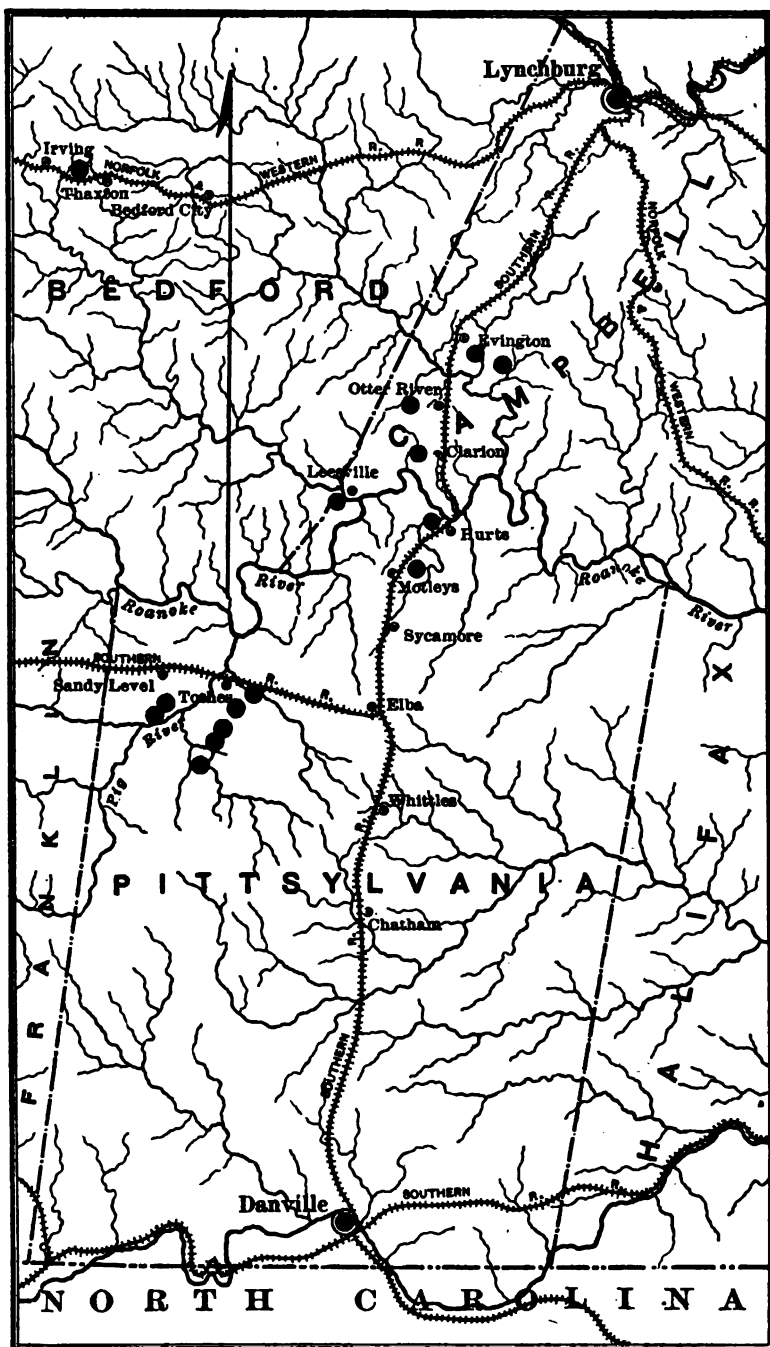


FIG. 17.—BARITE DEPOSITS OF THE BEDFORD-CAMPBELL-PITTSYLVANIA COUNTIES AREA, VIRGINIA. Scale, 1 in. = 7.5 m. approximately.

The barite occurs as irregular lense-like bodies or pockets replacing crystalline limestone associated with micaceous and quartzose schists, and as nodules and lumps in the residual clays, which are limonitic or magnaniferous and have been used in the manufacture of natural mineral pigments. Besides calcite and silicate minerals of the inclosing rocks, the associated minerals are galena, pyrite, and less sphalerite and chalcopyrite, in the unaltered deposits; and manganese oxide and iron oxide in the residual deposits. These impurities are often developed only sparingly and are not noticeable in the best grades of the barite. Fig. 16, an ideal section

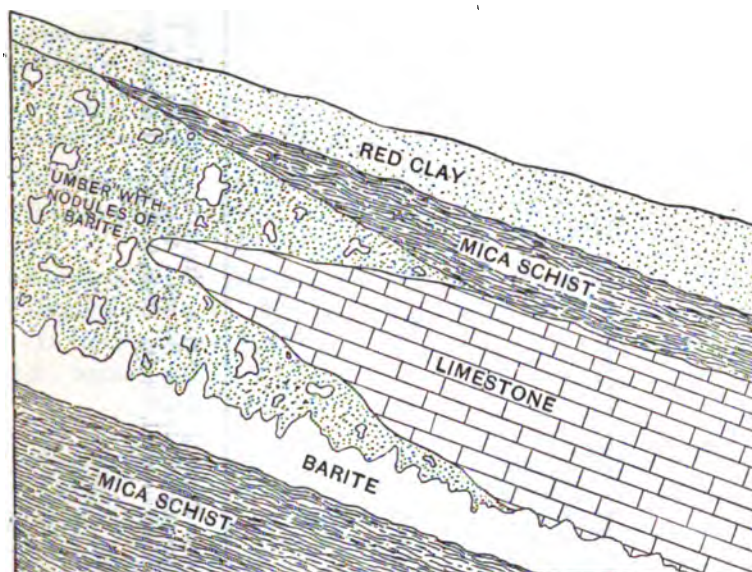


FIG. 18.—IDEAL SECTION IN BENNETT BARYTES MINE, PITTSYLVANIA COUNTY, VIRGINIA.

in the Bennett mine, Pittsylvania County, illustrates the mode of occurrence of the mineral in Campbell and Pittsylvania counties.

A sample of the black manganiferous clay collected from the Bennett mine, Pittsylvania County, by the senior writer and analyzed by W. B. Ellett, gave:

	Per Cent.
Insoluble residue.....	14.20
Alumina.....	4.96
Ferric oxide.....	32.40
Manganous oxide.....	19.49
Lime.....	2.06
Magnesia.....	trace
Barium oxide.....	trace
Copper.....	trace

The composition of the associated crystalline limestones is shown in the analyses below, made by W. B. Ellett on samples collected by the senior writer.

	1 Per Cent.	2 Per Cent.	3 Per Cent.
Insoluble matter.....	1.66	0.87	1.10
Alumina }	0.24	0.30	0.96
Iron oxide }			
Barium sulphate.....	0.62	0.65	1.62
Calcium carbonate.....	89.36	93.33	91.07
Magnesium carbonate.....	6.61	2.82	3.73
Copper sulphide.....	trace	trace	0.36

1 and 2. White crystalline limestone from the Hewitt mine, Campbell County.

3. White and pink crystalline limestone from the Ramsay mine, Pittsylvania County.

The barite is won chiefly by shafts and drifts, and open cuts and pits. The greatest depth reached is 160 ft., at the Hewitt mine in Campbell County.

Barite is found in association with similar rocks in other counties of the Piedmont province, but has not been mined to any considerable extent. In the western part of Bedford County, about 3 miles northwest of Thaxton, barite has been mined from a fracture-filled deposit in a coarse

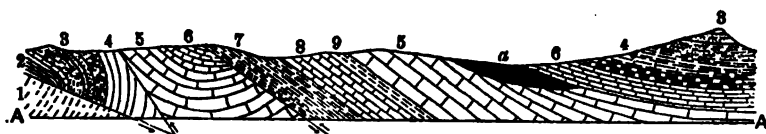


FIG. 20.—STRUCTURE SECTION ALONG LINE A-A OF FIG. 19, SHOWING STRUCTURAL RELATIONS OF THE BARITE AND ROCKS. *a* IS BARITE. ADAPTED FROM Tazewell Folio, U. S. Geological Survey.

granite. The mineral is crystalline, white to blue-gray in color, and in places contains much disseminated galena in small grains and occasional sphalerite. In Louisa County coarsely crystallized barite has been mined 3 miles south of east from Lindsay. It occurs in veins in schists and is remarkably free from impurities other than the usual discoloration from red iron oxide and some quartz. Several promising deposits of barite in the vicinity of Bealeton, Culpeper County, have been opened but no shipments have been made.

In the Valley region of southwest Virginia, in Tazewell, Wythe, Russell, and Smyth counties, barite occurs in the Cambrian and Ordovician limestones and their residual clays. The mineral occurs in the limestone as replacements and veins associated with sphalerite, galena, pyrite, and some fluorite. In Tazewell and Russell counties the common associates are limonite, chert, and calcite, with some siderite, and occasional fluorite.

Barite has been extensively mined near Marion, Smyth County, and extensive deposits are found in Russell and Tazewell counties, chiefly along the southern slope of Kent Ridge and its prolongation northeast and southwest, along the valley of Clinch River, extending from near North Tazewell to near Lebanon, a distance of more than 30 miles. The mineral has been mined at several points along this belt, the principal mines being

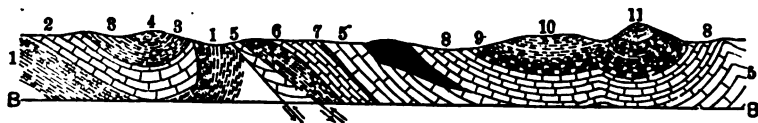


FIG. 21.—STRUCTURE SECTION ALONG LINE B-B OF FIG. 19, SHOWING STRUCTURAL RELATIONS OF THE BARITE AND ROCKS. BLACK IS BARITE. ADAPTED FROM *Tazewell Folio*, U. S. Geological Survey.

near North Tazewell, 3 miles south of Richlands, 3 miles from Honaker on the Clinch River, and at the southwestern end of the belt near Lebanon. The barite is found in the upper part of the Knox dolomite. Figs. 17, 18, and 19 show the structural relations of the Knox dolomite and the adjacent rocks on the northwest and southeast, near Sword Creek and Richlands, in Tazewell County.

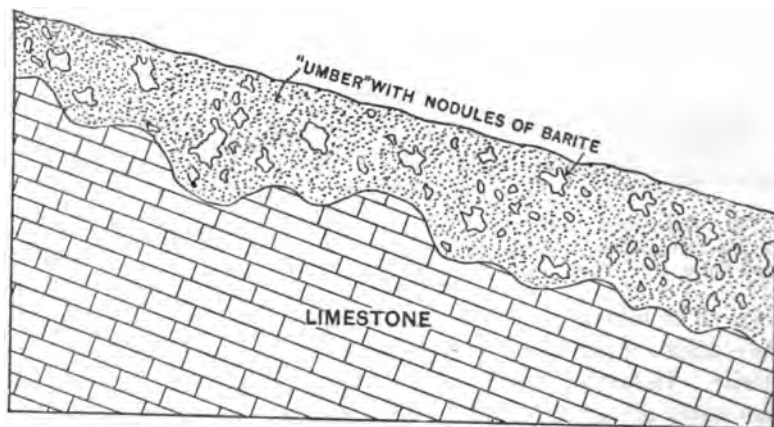


FIG. 22.—PRINCIPAL MODE OF OCCURRENCE OF BARITE IN RUSSELL AND TAZEVELL COUNTIES, VIRGINIA.

The barite occurs as lumps of varying shapes and sizes assembled in the residual clays of the limestone, and in pocket form and vein-like bodies filling spaces in the limestone, and in part as replacements of the limestone. Fig. 20 illustrates the residual mode of occurrence of the barite. The mineral is crystalline, of good white quality, and in most places is practically free from impurities. The greatest depth reached in mining is 103 ft., at the mines of the Pittsburgh Baryta & Milling Corporation on

the northeast end of the belt. Most of the mining done has been for the barite nodules in the residual clays, won from shallow open pits and cuts. Some hard-rock mining in the limestone has been done in places.

MINING METHODS

Mining of barite in the Appalachian States is chiefly surface work, with no really deep mining in any part of the region. Thus far the mineral has been won chiefly by open cuts, shallow shafts, and pits, which rarely exceed 40 ft. in depth, usually much less. This method of winning the mineral is especially applicable to the occurrence of barite in loose nodular masses of different sizes and shapes irregularly assembled in the residual clays.

In some districts, especially the Piedmont region of Virginia and central Kentucky, shafts ranging from 50 to 400 ft. in depth have been sunk. In Piedmont, Virginia, the barite is won by vertical timbered shafts and drifts which follow the mineral bodies, and the greatest depth yet reached in barite mining is less than 200 ft. In the central Kentucky region a depth of about 400 ft. has been reached but the shafts have been chiefly for prospecting. Tunneling has been practiced to some extent in this region in the high cliffs bordering the Kentucky River, where the veins are exposed for a vertical distance of 300 to 500 ft.

Blasting is necessary for breaking down the barite in the vein and limestone replacement occurrences of the mineral.

DEVELOPMENT AND TRADE CONDITIONS

The majority of the deposits of barite are so scattered and many of them so small that the problems connected with their operation are those of handling the mineral and getting it to market rather than of mining. However, there is room for much improvement in the methods of mining barite, as is evidenced by the large percentage of loss in the working of deposits that occur in residual clays. This loss, which in some cases amounts to as much as 30 per cent., is from the small lumps and grains recoverable by washing and jigging but too small for cutting and sorting by hand. A large number of the deposits are located at a distance from railway transportation, as well as being far removed from mills, and while this is a disadvantage that obtains in many instances, still many deposits occur which would be worked were not the freight rate the main deterrent. On the other hand, the problem that confronts the manufacturer at times is the difficulty of accumulating sufficient barite to keep the mill running continuously. The reason why the raw material is not always available is due to the fact that the barite producer lacks, because of decreased profits, the same incentive that actuates the manufacturer. Moreover, the foreign producer has found that he can deliver his product to the larger

city markets, like New York, Philadelphia, Baltimore, and others, cheaper than the domestic producer can deliver by rail.

Where barite occurs in residual clays the best way to handle it with a sufficient quantity of ore in sight would be with a steam shovel and suitable washers, but since few, if any, deposits are large enough to warrant the installation of such a plant and water is not everywhere plentiful the slower methods of mining, cobbing, and rocking must be employed. However, by employing energetic methods resulting in cheaper mining and the winning of a large percentage of the mineral that goes now to the dump heap in shape of small lumps and grains, the barite operators could increase considerably their present margin of profit, and thus operate profitably a large number of the deposits now idle.

METHODS OF PREPARATION

The preliminary treatment of barite for the market will vary according to the character of the mineral. In some localities the barite contains little or no impurities, in others the amount and character of the impurities vary greatly. The minerals associated with barite vary locally, but, broadly speaking, they vary in accordance with the mode of occurrence, whether residuary, or as veins and replacements. The more commonly occurring minerals are iron oxide, manganese oxide, limestone including calcite, and silica in the form of both quartz and chert, in the residual deposits; and the metallic sulphides, especially galena and sphalerite, fluorite, and calcite, in the vein and replacement deposits.

For the removal of impurities from the merchantable barite the principal steps in the preparation for market include hand cobbing, sorting or grading, washing, and crushing. For ground barite, bleaching, drying, and grinding are necessary. Such associated minerals as galena, quartz, calcite, and limonite, can be removed largely by hand cobbing during mining, but if these are inclosed in the barite, crushing and washing may often serve to cleanse it. Jigging is often found to be advantageous, especially if the iron oxide is in the form of scales or crusts, and it may be successfully employed for removing impurities of light weight, such as chert, limestone, etc. After the mineral has been hand cobbled at the mine into the several grades, it is shipped to the mills for further cleansing by crushing and washing to remove as much of the impurities as possible.

Much of the barite mined in the Appalachian States is won by open-cut work in residual clays derived from limestones; hence, the barite nodules are usually more or less stained with iron oxide and coated with clay. A method often practiced for freeing the mineral of these, especially the clay, is to assemble the nodules and lumps of the mineral in piles exposed to the weather, after which it is better prepared for hand cobbing and is given such additional cleansing as necessary prior to shipping to

the mills. In some cases washing the mineral in log washers is practiced, but this is not always feasible, since an ample water supply may not be available, and also loss of the fine material is likely to result from this treatment.

In some localities in the Appalachian States the washers installed have been crude and the management poor, hence poor results have been obtained. When feasible, however, log washing and jigging would yield a product vastly superior to that produced by sun-drying methods. Washing will not entirely cleanse the ore in all cases, especially when the nodules and lumps are coated with a heavy crust of iron oxide, as is often the case with the residual deposits, but it will appreciably reduce the iron content and correspondingly increase the price demanded.

Milling

Crude barite is treated by a number of different processes preparatory to a considerable range of uses. The equipment of the mill will depend upon whether the object be to prepare and market a ground and purified product, or to carry the treatment further by employing the comminuted material in the manufacture of compounds into which barium enters as an essential ingredient.

The barite as mined may be treated by either the wet or the dry process, the object in both cases being to reduce it to an impalpable powder and eliminate impurities, the two most difficult and deleterious being calcium carbonate and iron oxide. The former reacts injuriously with the acids of paints and the latter is objectionable because of its color.

In case the wet process is employed, suitable crushing, pulverizing, and jigging machinery is installed and the barite is kept in contact with water from the beginning to the completion of the treatment. When it is sought, however, to prepare the ground and purified product by the so-called dry method, the dry material is first passed through crushers and then cleansed as far as possible by water in log washers. The log washer, however, only partly eliminates such impurities as galena, sphalerite, iron oxides, calcite, fluorite, etc., hence it becomes necessary to bleach the material in an acid solution.

An outline of the complete process as followed by a thoroughly equipped mill has been described by Burchard as follows:²⁰

"The crude material is ground in slip mills having granite grinders and granite bases. Water is fed into these mills and the ground material is floated over the top of the tanks, after which it is pumped into funnel-shaped separators. The contents of the separators are agitated by flowing water and the coarser rejected material is drawn off at the bottom of the funnel and returns to the slip mills, while the finer material floats off at the top of the separators. This material next descends to

²⁰ Burchard, E. F.: *Mineral Resources*, U. S. Geological Survey, pt. ii, p. 687 (1907).

settling tanks, and after forming a sludge is drawn off into bleaching tanks. The bleaching tanks are built of concrete lined with refractory tile. Bleaching is accomplished by the addition of measured weights of sulphuric acid to the sludge and the agitation of the mass to secure thorough mixture. The acid reacts on the iron oxide and lime present, forming ferrous sulphate and calcium sulphate. The iron salt being soluble and the calcium salt partially soluble, besides having a lower specific gravity than the pure barytes, these substances, together with the excess of sulphuric acid, are removable in the further washing process to which the material is subjected. For this next washing the material is pumped into washers which employ the float-separation process. Next, the bleached barytes passes to settling tanks, after which it is dried by being spread thinly on the surface of a rotating hot drum. From the hot drum the dried material falls or is brushed off and carried to Williams mills, where it is pulverized, screened, and finally sacked by machine. The essential difference between this process and the others mentioned above is in the fact that the material is first reduced to a fine condition before bleaching, thereby bringing the sulphuric acid intimately into contact with all portions of the barytes."

Bleaching

Nearly all crude barite requires bleaching before the requisite degree of whiteness can be secured for the finished product. In order to free barite from iron stain, bleaching with sulphuric acid is necessary. The mineral is first crushed and washed, when it is ready for bleaching. The degree of fineness to which the mineral is crushed will depend upon the amount and character of iron present. In the Appalachian States, the barite is usually crushed to a size varying from that of a pea to $\frac{1}{2}$ in., according to the amount of iron oxide, and whether it is in the form of scales or crusts, or as a stain. Jigging is necessary in some cases to remove the iron stain. Higgins²⁷ has described the bleaching process as follows:

"The crushed mineral is bleached in circular wooden tanks, lined with sheet lead. Each tank is provided with a 2-in. lead pipe, coiled in the bottom, closed at the end, and provided with small perforations, 7 in. apart. Steam escaping from the perforations agitates and heats the bleaching solution. This is sulphuric acid diluted to about 20 or 30° B. The mineral is charged into the tanks to a depth of about 3 ft., after which the acid solution is run in and steam is turned on. The time required to bleach the mineral varies from 6 to 80 hours, depending on the iron content. It is the usual practice to leave the steam on continuously, although good results may be obtained by cutting it off for half an hour at intervals of one hour."

"A tank 4.5 ft. high and 8 ft. in diameter is a convenient size for bleaching, containing, when charged to a depth of 3.5 ft., approximately 14 tons. It should be made of stout, well-seasoned wood, preferably cypress, should be well braced on the outside, and lined on the inside with heavy sheet lead. Connections should be provided so that either steam or water may be supplied through the perforations in the lead pipes. These perforations are best located at an angle of 45° off the vertical rather than directly on top, in order to prevent clogging of the pipes with fines.

"One of the greatest items of expense in the process is the steam for heating and agitating the bath. This may be greatly reduced by use of an ordinary injector,

²⁷ Higgins, Edwin: *The Mineral Industry*, vol. xiii, p. 34 (1904).

using a mixture of air and steam instead of steam alone. The amount of air can easily be regulated, so that the acid will not be too much cooled. The temperature may be kept at 200° F.; better agitation will result; the acid will suffer less dilution from condensing steam; and the steam consumption will be reduced one-half."

Should manganese dioxide be present in the barite it is not removed by the bleaching process. It becomes necessary then to grind the barite containing manganese oxide to a paste on a 40-mesh screen, and mix it in the proper proportion with sodium nitrate, salt, and sulphuric acid. The mixture is heated in a specially constructed furnace, which converts the iron and manganese into soluble chlorides that are readily removed by washing, the barite being collected in a series of settling tanks (usually three in number).

After bleaching, the barite is reduced to the size of fine sand by rolls and then to an impalpable powder by buhrstones, after which it is ready to be packed for shipment.

Drying

In drying bleached barite regard should be had for an even temperature, in order that none be sent to the mill imperfectly dried, otherwise it may "sweat" before going to the buhrs, which will yield a finished product that is off color. If mechanical conveyors are used for delivering the barite, care should be taken that the material does not come in contact with oil.

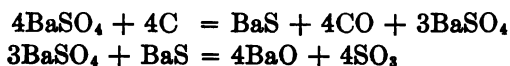
Grinding

For use in the paint and rubber trade the barite must be finely ground. This is accomplished almost entirely by buhrs, since no other method of grinding yet tried has proved so satisfactory in reducing the material to the required degree of fineness.

BARIUM COMPOUNDS

Barium enters as a principal constituent into a number of artificial compounds, which have a wide range of uses. Some of the more important compounds include barium oxide, barium chloride, barium carbonate, lithopone or lithophone, and blanc fixe. These are briefly described below in the order named.

Barium Oxide.—This compound is prepared according to the Bradley and Jacobs process by treating a mixture of barite and carbon in the electric furnace. The following reactions which occur simultaneously are reported:



Barium hydrate, used chiefly in the beet-sugar industry, may be obtained from the resultant mixture by treating it with hot water. In France, barium dioxide is used in the production of hydrogen dioxide, which finds an extensive use as an antiseptic and bleaching agent.

Barium Chloride.—As practiced by a Southern plant, this compound is prepared by heating in an 80-ft. rotary cement kiln a mixture of barite and coal, which after reduction yields barium sulphide and barium carbonate. The resulting mixture is leached with water, which removes the barium sulphide, and the residue treated with hydrochloric acid. The sulphide solution is treated with hydrochloric acid, filtered, purified, and evaporated to the point of crystallization.

Barium chloride has extensive use in the color industry and in the manufacture of wall paper. The fused chloride is used to a limited extent in tempering tool steel.

Barium carbonate (artificial), imported as witherite, is used to neutralize the soluble sulphates in those clays employed for the manufacture of terra cotta, to a limited extent in the manufacture of glass, and as a chemical reagent.

The Seurre process²⁸ for the manufacture of barium carbonate from barite consists in heating to the fusion point with constant stirring equal parts of pulverized barite and calcium chloride with 5 or 6 per cent. of powdered charcoal in a crucible or furnace. The mass when leached in water yields a solution of barium chloride and an insoluble residue of calcium sulphate. Barium carbonate is precipitated from the filtered chloride solution by the addition of a 50 per cent. solution of ammonium carbonate, filtered and washed. The solution containing ammonium chloride may be treated with chalk, and the ammonium carbonate and calcium chloride returned to the process.

Lithopone.—Originally lithopone consisted of a simple mixture of barium sulphate and zinc sulphide, but later this was improved upon and the two were obtained as a double precipitate. At present the product may consist of barium sulphate with either zinc sulphide or zinc oxide or both. In the trade lithopone has been known under different names, such as Charlton White, Orr's White, Griffith's White, Becton White, etc. The best grades of lithopone are white, but inferior grades may be some shade of gray, brown, or yellow, due to the presence of such impurities as carbon, iron, etc.

The method usually employed in the precipitation of lithopone consists in mixing solutions of barium sulphide and zinc sulphate in the proportion of 51 parts of the former to 49 parts of the latter, which results in the precipitation of barium sulphate and zinc sulphide. The barium sulphide is prepared by heating a mixture of crude barite (four parts by weight) and fine coal (one part) to bright red heat in a reducing atmos-

²⁸ *The Mineral Industry*, vol. xiv, p. 45 (1905).

phere. "In Germany lithophone is made largely from zinc sulphate obtained directly from ore at the works in the lower Harz, but in the United States the zinc sulphate is chiefly obtained by dissolving scrap zinc, or zinkiferous by-products, in sulphuric acid. At one works scrap brass is the source of the zinc employed."²⁹

Blanc Fixe is chemically pure barium sulphate, and is precipitated from a solution of barium sulphide either by sulphuric acid or by salt cake. When salt cake is used sodium sulphide is formed, for which there is a limited market. The artificial barium sulphate ("blanc fixe") is pure and free from grit, and is employed chiefly in the manufacture of the finer grades of white paint, oil cloth, and paper. Large quantities of the product used in the United States are imported from Germany.

Until within the past few years this product was confined to the imported lithopone, chiefly from England and Germany, but an important home industry has now been established, and in 1908 at least nine firms in the United States were manufacturing lithopone.

MANUFACTURERS OF BARIUM PRODUCTS

A table giving the names and addresses of the consumers of crude barite in the United States was published in *The Mineral Industry*, vols. xvii to xx (see bibliography at end of this paper for page references).

USES

Much the greater part of the production of barite is consumed as an inert pigment in the manufacture of mixed paints, the ground mineral being used both in its natural and in its artificial forms for this purpose. For this use the mineral must be finely ground, washed, bleached with acid, and floated. As a pigment, barite is stable in the presence of acids, alkalies, and gases, since it undergoes no chemical action and is inert as to color. Its crystalline nature renders it unsatisfactory as a pigment if used alone, and only moderate percentages should be used in mixed paints in order to secure the best results. Barite is considered a good base upon which to precipitate other colors, and it is used to dilute other pigments which tend to destroy the oil. Moreover, it gives "tooth"³⁰ to a coat of paint, so that a second or third coat will adhere.

According to Gardner and Heckel, probably the largest amount of barite is used in the production of the artificial pigment lithopone, which

²⁹ *The Mineral Industry*, vol. xvi, p. 95 (1907).

³⁰ "'Tooth' is an expression used to describe the brushing action of a paint, which indicates that the paint is being taken up by the wood or other surface to which it is applied." *Bulletin No. 93*, Sept., 1914, p. 2355.

consists of 70 parts of barium sulphate and 30 parts of zinc sulphide. They say:²¹

"An enormous tonnage of this pigment is at the present time being used in the manufacture of interior flat wall paints of the oil type, such paints having almost entirely supplanted the use of corroded white lead which was at one time used for interior painting. The more dense nature, lower price and sanitary value of lithopone were responsible for this change. Lithopone is, moreover, used to a large extent in the manufacture of oil cloth, shade cloth and linoleum. Its whiteness makes it of particular value for this purpose. Barium sulphate is also used of itself in admixture with other pigments in the manufacture of prepared paints. The chemically precipitated form of barium sulphate, which is generally called *blanc fixe*, is also used for this purpose."

These writers conclude that there should be no technical objection to the use of barite in paints. They say:²²

"The only question is an economic one, and involves two considerations: The first is that barytes and other inert pigments shall not be sold except under their true names and shall not be used as adulterants, using that term in the sense given it by the American Society for Testing Materials, as follows: 'Adulterant—A substance partially substituted for another.' The other consideration is that the amount of barytes shall be such that it will not impair the opacity of the paint under the conditions of application, which implies that three coats shall effectively hide the underlying surface to which it is applied."

The uses of some of the more important barium salts have already been stated. Barium is also used in the manufacture of glass, especially in rolled glass, hollow ware, crystal and table glass, and in such special glasses as the Jena phosphate crown glass, which is reported to contain 28 per cent. BaO. Other uses of barite are in the manufacture of rubber, wall paper, asbestos cement, tanning leather, refining sugar, pottery glazes, enameling iron and oil cloth, poker chips, coating canvas ham sacks, and in the preparation of fertilizers, boiler compounds, insecticides, hydrogen peroxide, and artificial driftwood salts.

BIBLIOGRAPHY

The subjoined list is not a complete bibliography of barite, but it includes the most important recent contributions to the literature. Other references not included in the bibliography are cited under individual States.

Alzugaray, Baxeres de: Composition, Distribution, and Production of Barytes, *Mining and Engineering World*, vol. xxxvii, No. 1, p. 17 (July 6, 1912).

Andrée, K.: Ueber ein bemerkenswertes Vorkommen von Schwerspat auf dem Rosenhofe bei Clausthal, *Zeitschrift für praktische Geologie*, vol. xvi, p. 281 (July, 1908).

²¹ Gardner, H. A., and Heckel, G. B.: Barytes as a Paint Pigment, *Bulletin No.* 93, Sept., 1914, p. 2353.

²² *Idem*, p. 2355.

Bryant, F. C.: Barytes Industry of Cole County, Mo., *Engineering and Mining Journal*, vol. xcv, No. 6, p. 317 (Feb. 8, 1913).

Burchard, E. F.: A Barite Deposit near Wrangell, Alaska, *Bulletin No. 592, U. S. Geological Survey*, pp. 109 to 117 (1914).

Canadian Mining Journal: Nova Scotian barite, vol. xxxiii, No. 18, pp. 661, 662 (Sept. 15, 1912); No. 19, p. 678 (Oct. 1, 1912).

Catlett, Charles: Barite Associated with Iron Ore in [Pinar del Rio Province, Cuba, *Trans.*, xxxviii, 358, 359 (1907).

Engineering and Mining Journal: Barium Salts, vol. xcvi, No. 20, p. 1015 (May 20, 1911).

Fay, A. H.: Barytes in Tennessee, *Engineering and Mining Journal*, vol. lxxxvii, No. 2, p. 137 (Jan. 9, 1909).

Foos, F. Julius: Barytes Deposits of Kentucky, *Kentucky Geological Survey*, ser. iv, vol. i, pt. 1, pp. 441 to 588 (1913).

Frazier, Schuyler: Barytes, Occurrence and Methods of Preparation, *Chemical Engineer*, vol. xiii, No. 2, pp. 43 to 46 (Feb., 1911).

Gardner, H. A., and Heckel, G. B.: Barytes as a Pigment, *Bulletin No. 93*, Sept., 1914, pp. 2353 to 2355.

Grasty, J. S.: Barite Deposits of Kentucky, *The Tradesman*, July 3, 1913, pp. 33, 34.

——— Barite Deposits of Alabama, *The Tradesman*, July 17, 1913, pp. 35, 36.

Hayes, C. W., and Phalen, W. C.: A Commercial Occurrence of Barite near Cartersville, Ga., *Bulletin No. 340, U. S. Geological Survey*, pp. 458 to 462 (1908).

Henegan, H. B.: Barite Deposits in the Sweetwater District, Tenn., *The Resources of Tennessee*, State Geological Survey, vol. ii, No. 11, pp. 424 to 429 (Nov., 1912).

Higgins, Edwin: Barytes and Its Preparation for the Market, *Engineering News*, vol. liii, No. 8, pp. 196 to 198 (Feb. 23, 1905).

Hurst, George H.: *Painters' Colours*, pp. 74, 89, 142 (1896). Methods of washing barite, clay, and ocher.

Hutchinson, W. S.: Barytes Deposits at Five Islands, Nova Scotia, *Engineering and Mining Journal*, vol. lxxxiv, No. 18, pp. 825, 826 (Nov. 2, 1907).

Jennison, F. H.: Lake Bases, Their Composition and Uses, *Oil, Paint and Drug Reporter*, vol. lxxxi, No. 11, pp. 27, 28 (1912).

Jones, J. C.: Efflorescence of Brick: *Studies from School of Ceramics, University of Illinois*, vol. iv, No. 4 (Oct. 15, 1906), 22 pp.

Judd, E. K.: The Barytes Industry of the South, *Engineering and Mining Journal*, vol. lxxxiii, No. 16, pp. 751 to 753 (April 20, 1907).

——— A Barytes Grinding Plant, *Engineering and Mining Journal*, vol. lxxxiii, No. 21, pp. 996, 997 (May 25, 1907).

Lakes, Arthur: A New and Large Deposit of Barite in Idaho, *Mining Reporter*, vol. liv, No. 27, p. 162 (Aug. 16, 1906).

Miller, A. M.: The Lead and Zinc Rocks of Central Kentucky, with notes on the mineral veins, *Bulletin No. 2, Kentucky Geological Survey*, pp. 24 to 35 (1905).

Mineral Industry, The: vol. ii, pp. 53 to 56; vol. viii, pp. 55, 56; vol. x, pp. 58 to 61; vol. xiii, pp. 32 to 38; vol. xiv, pp. 42 to 45; vol. xv, pp. 65 to 74; vol. xvi, pp. 88 to 96; vol. xvii, pp. 72 to 78; vol. xviii, pp. 60 to 66; vol. xix, pp. 68 to 76; vol. xx, pp. 79 to 91; vol. xxi, pp. 77 to 85; vol. xxii, pp. 42 to 46 (1893 to 1913).

Mineral Resources of the United States, U. S. Geol. Survey: 1901, pp. 915 to 919; 1902, pp. 945 to 948; 1903, pp. 1089 to 1093; 1904, pp. 1095 to 1102; 1905, pp. 1145, 1146; 1906, pp. 1109 to 1114; 1907, Pt. II, pp. 685 to 695; 1908, Pt. II, pp. 669 to 673; 1909, Pt. II, pp. 697 to 700; 1910, Pt. II, pp. 799 to 802; 1911, Pt. II, pp. 965 to 970; 1912, Pt. II, pp. 955 to 959; 1913, Pt. II, pp. 165 to 174.

Norwood, C. J.: Kentucky Barytes and Calcite, *Engineering and Mining Journal*, vol. xciii, No. 9, p. 458 (Mar. 2, 1912).

Poole, H. S.: The Barytes Deposits of Lake Ainslee and North Cheticamp, N. S., with notes on the production, manufacture, and uses of barytes in Canada, *Bulletin No. 953, Geological Survey of Canada* (1907). 43 pp.

Singewald, J. T., Jr.: The Barite Deposits at Meggen on the Lenne, *The Tradesman*, Apr. 3, 1913, pp. 32, 33.

Steel, A. A.: The Geology and Preparation of Barite in Washington County, Mo., *Trans.*, xl, 711 to 743 (1909).

——— The Geology and Mining of Barite in Missouri, *Mining World*, vol. xxxii, No. 9, pp. 463 to 467 (Feb. 26, 1910).

Stose, G. W.: Barite in Southern Pennsylvania, *Bulletin No. 225, U. S. Geological Survey*, pp. 515, 516 (1904).

Watson, T. L.: Geology of the Virginia Barite-Deposits, *Trans.*, xxxviii, 710 to 733 (1907); *The Tradesman*, Mar. 20, 1913, pp. 38 to 41; Mar. 27, 1913, pp. 31 to 33.

——— Fluorite and Barite in Tennessee, *Trans.*, xxxvii, 890 (1906).

Watson, T. L., and Grasty, J. S. The Barytes Deposits of Tennessee, *The Tradesman*, May 1, 1913, pp. 34 to 38.

——— The Geology and Barite Deposits of the Cartersville, Ga., District, *The Tradesman*, May 22, 1913, pp. 35 to 37.

Wittich, L. L.: Barytes in Missouri, *Mines and Minerals*, vol. xxxiii, pp. 95 to 97 (1912).

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

White-Burning Clays of the Southern Appalachian States

BY JOEL H. WATKINS, WASHINGTON, D. C.

(New York Meeting, February, 1915)

THE terms kaolin, china clay, ball clay, and paper clay are more or less loosely and interchangeably applied to a large class of white-burning clays. These clays are made up chiefly of hydrous amorphous (colloidal) aluminum silicates with variable amounts of free silica and other impurities in small quantities. Occasionally also the term kaolinite is wrongly applied to a white-burning clay, for kaolinite is a mineral of definite chemical composition and definite crystalline form which occurs but sparingly in nature.

It is not the purpose of this paper, however, to define or classify clays, or to enter into a discussion of the chemistry of clays. It is rather a general description of the occurrence and methods of mining of a kindred group of clays, and a discussion of their future economic aspect.

In the Southern Appalachian States, the mining of white-burning clays has in recent years become an important industry. Although these clays are of several distinct types, which occur in as many localities under different geologic conditions, they are for the most part primarily of the same origin. They are essentially the same in elementary constituents, and are largely consumed in the same industrial arts. From the standpoint of a mining engineer, therefore, and for convenience in comparison and correlation, it should be interesting to discuss them under one heading.

KINDS OF CLAY

The most important types of clay which are now being actively mined in the Southern Appalachian States and with which this paper is particularly concerned are the North Carolina, Virginia, and Georgia kaolins or china clays (residual); the South Carolina and Georgia kaolins or paper clays (sedimentary); and the Florida and Tennessee plastic kaolins or ball clays (sedimentary). Other varieties which are interesting but which are as yet of little importance are the pocket deposits of white clay in Augusta County, Virginia, the bauxitic kaolins of Tennessee,

Georgia, and Alabama, and the halloysite deposits of Tennessee and Georgia. (Fig. 1.)



- RESIDUAL KAOLIN MINES, ■ SEDIMENTARY KAOLIN MINES,
- ◆ FLORIDA BALL CLAY MINES, ▲ TENNESSEE BALL CLAY MINES.

FIG. 1.—MAP SHOWING GEOGRAPHIC DISTRIBUTION OF CLAY MINES IN THE SOUTHERN APPALACHIAN STATES.

I. RESIDUAL KAOLINS OR CHINA CLAYS

Origin.—The term kaolin is probably most correctly used when applied to white-burning residual clays which are derived directly from the

decomposition of feldspars in place, and which, when pure, conform closely to the chemical composition of the mineral kaolinite. The process of decomposition of feldspars, resulting in the formation of kaolin, is termed kaolinization. When kaolinization takes place to sufficient depth in highly feldspathic bodies of good dimensions, it often results in the formation of valuable deposits of china or paper clay. True pegmatite dikes probably afford the best examples of deposits of this character.

Distribution.—Pegmatite dikes occur abundantly throughout the crystalline area of the Southern Appalachian States. Workable deposits of kaolin are developed in Virginia, North Carolina, and Georgia.

North Carolina is at present by far the most important of the States which have produced residual kaolin. Pegmatite bodies of large and small size are so numerous over portions of Yancey, Mitchell, Jackson, Swain, and Macon counties that one might say the country rock has simply been saturated with granite juice. Mining is most active in these counties.

In Virginia, kaolin has been produced from pegmatites at only two localities: namely, about $3\frac{1}{2}$ miles southeast of Henry, in Henry County, and about 8 miles northwest of Arrington, in Nelson County. Neither of these plants is being operated at this time. Other deposits in Virginia which have been developed to some extent are near Motley in Pittsylvania County and near Forest Depot in Bedford County.

In Georgia, kaolinized pegmatities have been prospected in Paulding, Pickens, Rabun, White and other counties of the Piedmont region, but, so far as known, there has been no production of residual kaolin in the State.

Geologic Relations.—In North Carolina, the pegmatite dikes that have proved to be of chief commercial importance are principally confined to the rough and mountainous region of the western part of the State. In their distribution they favor no particular topography, but are found alike on the slopes of high mountains and in the intervening valleys. They are intruded into granites, gneisses, and schists, but are more highly developed in the gneisses and schists, which offer less resistance along lines of rock cleavage. They may vary in width from a few inches up to several hundred feet, and in length from a few feet up to several miles. They are lenticular bodies which "swell" and "pinch" both laterally and vertically, the perpendicular dimension being more persistent. In the Appalachian States these dikes generally conform roughly to the foliation of the inclosing rocks, though they sometimes cut across the structure. In most cases the dikes are nearly vertical, though occasionally the dip is quite oblique. The inclosing rocks are pre-Cambrian in age, and of two distinct types, the Carolina gneiss and the Roan gneiss. The Carolina gneiss consists of a great series of mica-schist and garnet-

smaller plants in North Carolina, which was promoted on the prospects of deriving its kaolin from a dike about 8 ft. wide which had a dip of about 45°, after producing 50 tons of kaolin, was forced to suspend operations. This company would have been a monumental failure had not one of its employees about that time discovered, 1½ miles from the plant, a kaolinized pegmatite about 200 ft. wide on ground which had been traversed repeatedly by other prospectors. The writer had the privilege of making a report on this property for the owners, and from 18 samples carefully taken from a number of shafts (Fig. 3) and test pits obtained the following percentage recovery of kaolin after washing. Samples Nos. 1 to 6 were taken at various depths in a 62-ft. shaft and Nos. 7 to 18 from test pits.

Sample No.	Sampling Depth, feet	Kaolin, Per Cent.
1	15	28.81
2	25	21.95
3	35	44.82
4	45	19.15
5	55	15.79
6	62	10.52
7	10	24.44
8	15	21.43
9	15	28.90
10	6	30.19
11	10	21.74
12	15	30.95
13	15	28.00
14	15	44.60
15	15	46.66
16	10	40.00
17	25	22.38
18	10	10.35

The average percentage recovery of kaolin from the 18 samples washed was 27.26, which is somewhat higher than should be expected from the actual mining and washing of the entire kaolinized portion. This deposit will probably prove to be one of the most extensive yet developed in North Carolina.

Mining and Preparation.—The mining and treatment of residual kaolin in the Appalachian Mountain region has been so thoroughly discussed by Watts in a recent bulletin issued by the U. S. Bureau of Mines that only a summary of the essential details will be outlined in this paper. Emphasis will also be given some of the points advocated by Mr. Watts in regard to the future development of these deposits.

Where open-cut mining is possible, it is, of course, always the easiest, and has first preference. The method of mining kaolin by open circular shafts, however, which is exclusively practiced in the case of residual deposits, is both unique and economic (Fig. 4). Since in the zone of



FIG. 3.—TYPICAL PROSPECT SHAFT, WESTERN NORTH CAROLINA RESIDUAL KAOLIN.



FIG. 4.—TYPICAL NORTH CAROLINA RESIDUAL KAOLIN MINE.

weathering the wall rock is decayed to about the same depth as the pegmatites, underground mining by the usual methods is both difficult and dangerous. Open-cut mining is always carried forward until slides from the walls become troublesome. The open cuts vary in depth with width of dike from less than 10 ft., in very narrow ones, up to 40 ft. in those 100 ft. wide and more. The smaller dikes are almost invariably richer in feldspar and consequently when weathered are richer in kaolin. Kaolinization in some instances reaches a depth of more than 100 ft., but the average depth of workable deposits is probably nearer 50 ft. Most dikes have a dip of more than 70° , but when the dip is less than 70° and the width is less than 20 ft. the problem of winning the kaolin becomes complicated. Ten feet is probably a minimum workable width for a pegmatite high in kaolin which has a nearly vertical dip, and a dike of such dimensions can be worked to a depth of 50 ft. with a shaft 13 ft. in diameter. If a dike 10 ft. wide, however, has a dip of less than 60° , the foot wall would soon be penetrated by a vertical shaft.

It has not been found practicable to undertake mining by open circular shafts on an incline. In the wider dikes, shafts are started where open-cut work is left off. When a shaft reaches the depth at which the percentage of recoverable kaolin is too low to be of value, another shaft is started a short distance from the first. The space between the shafts is generally robbed through holes made by removing timbers at different levels. The abandoned shafts are filled with waste from other shafts and cuts, while stoping between the shafts and removal of timbers advances as the shaft is filled. In this way, all of the kaolin is won and the same timbers can be used repeatedly. The shafts vary in diameter from 13 to 20 ft., and several men can work in the bottom of each shaft without being crowded. Ordinary derricks with mast and boom operated by drum hoists are in common use. Half-barrel buckets, which hold about 500 lb. of material, are lowered into the open shaft and filled while other buckets are being hoisted and dumped in flumes or on waste heaps.

With the exception of quartz veins and occasional masses of semi-kaolinized feldspar within the pegmatites, all of the material mined can be won with pick and shovel. Massive quartz veins often several feet thick generally form one of the walls of the pegmatites, though sometimes found within the dike itself. They are difficult to move, and as there is little or no market for quartz in this territory it is for the present thrown in with other waste materials. Sheet mica of marketable dimensions and such minerals as pitchblende, beryl, and gems are always saved and in some cases go a long way toward paying the running expenses of the mine. Accessory minerals are uncertain factors, however, and should never be counted in the valuation of a kaolin mine.

As the mines are always on high ground the kaolinized material can generally be flumed to the washing or refining plant. These plants

require a great deal of clear water and are generally located along streams and as near a railroad as possible. With sufficient fall and well-constructed flumes, it is possible to transport the material several miles to a washer.

The washing plants consist of a series of disintegrators, sand wheels, sand troughs, mica troughs, screens, settling tanks, agitators, and filter presses, which represent the different steps in separating kaolin slip from waste materials. After the refined kaolin is taken from the filter presses, it is dried either in open sheds, on steam tables, or in hot-air tunnels.

Analyses of North Carolina Residual Kaolins

(Analysis No. 1 by Ries, Nos. 2, 3, and 4 by Watts)

	1	2	3	4
SiO ₂	45.40	46.95	44.00	45.20
Al ₂ O ₃	39.34	37.24	40.79	38.45
Fe ₂ O ₃	1.92	0.40	0.11	0.45
TiO ₂		0.05	Trace	Trace
CaO.....	0.44	Trace	Trace	Trace
MgO.....	0.20	Trace	Trace	Trace
K ₂ O and Na ₂ O.....	0.52	0.73	0.62	0.65
H ₂ O.....	13.96	14.10	14.72	14.80
Total.....	101.78	99.47	100.24	99.55

II. SEDIMENTARY KAOLINS OR PAPER CLAYS

Distribution.—In the Cretaceous horizon of South Carolina and Georgia are extensive beds of white clay which have been actively mined at several points for a number of years. These Cretaceous beds are exposed in a narrow belt extending in a general northeast-southwest direction across the States. The principal counties in which important beds of kaolin have thus far been developed are Bibb, Twiggs, Wilkinson, Washington, Glascock, Jefferson, and Richmond counties, Georgia; and Aiken, Edgefield, Lexington, Richmond, and Kershaw counties, South Carolina. Mining is most active in the vicinity of Dry Branch, McIntyre, and Hephzibah, Georgia, and Aiken, Langley, and Bath, South Carolina. In all there are about 12 companies producing kaolin in the two States at this time.

Origin.—From a genetic standpoint these clays have been correctly termed sedimentary kaolins, since they represent an accumulation of finely divided particles of kaolin which have been deposited from suspension in water. The primary origin of these particles of kaolin is attributed to the residual clays in the crystalline area of the Southern Appalachian region. A large portion of this area is underlain with

highly feldspathic rocks, such as granites, syenites, pegmatites, aplites, or some of the finer-grained acid intrusives. The residual clays derived from these rocks contain more or less kaolinized feldspar, which for a long period of years has been gradually taken up by surface erosion and transported by streams to the sea. At some period during Cretaceous times, all of the conditions favorable to the concentration and deposition of these kaolin particles must have existed.

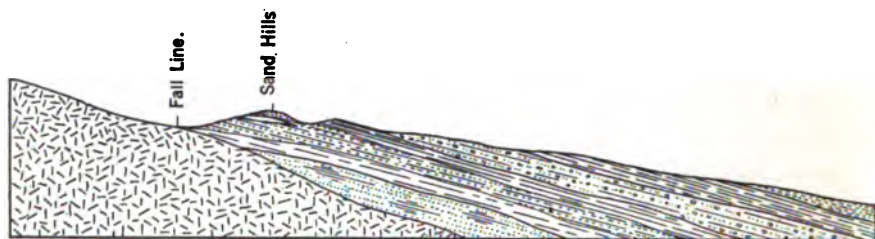


FIG. 5.—SECTION SHOWING RELATION BETWEEN COASTAL PLAIN SEDIMENTS AND UNDERLYING CRYSTALLINE ROCKS.

From a comparison of the analyses of these clays with analyses of the North Carolina residual kaolins, it will be seen that they are closely similar, the greatest variation being that the ferric oxide and titanium oxide are somewhat higher in sedimentary clays than in the residual clays. This similarity in chemical composition, together with the presence of coarse white quartz sand and small particles of white mica, is additional evidence that the bedded kaolins are accumulations of transported materials derived from the residual of feldspathic rocks.

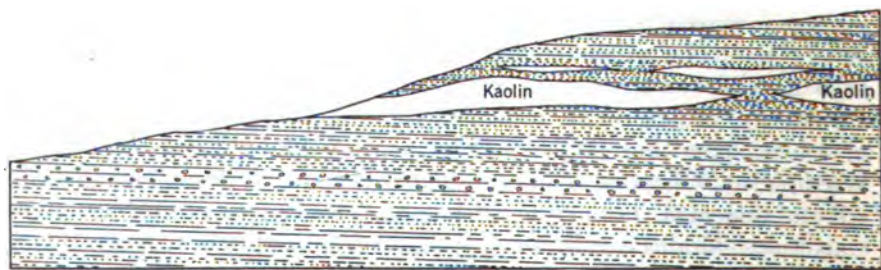


FIG. 6.—SECTION SHOWING GENERAL OUTLINE OF KAOLIN BEDS AND RELATION TO UNDERLYING AND OVERLYING SEDIMENTS.

Geologic Relations.—The lower Cretaceous beds in South Carolina and Georgia comprise a series of cross-bedded sands, clays, and gravels which rest directly upon the crystalline rocks of the Piedmont Plateau (Figs. 5 and 6). The irregular surface contact between the Cretaceous and the crystallines is commonly known as the "fall line." This line is marked by a series of sand hills through which streams passing from

the Piedmont Plateau to the Coastal Plain have cut small valleys, and the streams themselves often have considerable fall. The general elevation of the upland country of this narrow belt is between 400 and 600 ft. above sea level. In this division of the Cretaceous lying above the surface contact with the crystallines occur the great bedded deposits of white clay which are now so important to our domestic paper mills and pottery plants. These beds of clay are by no means continuous, but may vary in thickness from a few inches up to 35 ft. within a comparatively short distance. In one instance the writer has observed in a railroad cut as many as three distinct beds of very white clay, one above the other, which were not more than 4 in. at greatest thickness, and which in a short distance thinned out to the thickness of a knife blade. The contact of the clay with the underlying and overlying sands is generally quite distinct, but in places the sand seems to grade into the clay for a few inches. The plane marking the contact with the underlying beds is much more regular than that marking the contact with overlying beds. Within the thickness of a single bed the clays may vary greatly in color, though the change in color is generally abrupt in different parts of the bed. For example, in one place the writer observed the following changes in color from top to bottom of a bed: Deep purple, 9 in.; light pink, 12 in.; light yellow, 2 ft.; white, 10 ft.; blue-gray, 3 ft. The overburden, which may be from zero to 50 ft. in thickness, consists of a series of reddish cross-bedded sands, sandy clays, and gravel with occasional thin ledges of hard sandy brown iron oxide.

Prospecting and Developing.—Probably the larger portion of these extensive clay deposits has already been eroded away and transported by surface waters. There still remain, however, wide areas of this sand-hill country which may be prospected for kaolin beds. As is usually the case, only those deposits which have easy access to railroad transportation have been developed. The outcrops are generally exposed on the slopes of hills, as the sandy beds are very loose and erosion is rapid. The surface exposures, however, only show the presence of the material and do not afford a fair idea of the dimensions of the deposits. Auger drills can be easily driven through the soft sand-clay sediments, and the actual thickness and character of a deposit can be satisfactorily determined by this means in a surprisingly short time.

Systematic prospecting in this field, with the aid of an engineer's level and an auger drill, should develop other workable deposits of kaolin at a relatively small expenditure.

Mining and Preparation.—The mining of sedimentary kaolins to the casual observer would seem but a simple operation; moving a few feet of soft sandy overburden, then digging the snowy white clay from the banks with pick and shovel, and laying it on shelves to dry. If the deposit always had access to transportation, if the overburden were always but

a few feet, and if there were no rainfall, the operation might be a simple one indeed. The mine manager in this field, however, has his troubles, as well as those who have more complicated problems.

The kaolin is always first mined along the outcrop or where the overburden is least expensive to move. After the kaolin which is most easily won is exhausted, greater difficulties confront the manager in pushing his excavations further into the hill and moving the heavier overburden. Open-cut mining is universally practiced, as underground methods have been unsuccessful. The overlying materials are so soft that the timbering would have to be water-tight in order to keep out the sand and iron-stained silt in rainy weather. The writer on one of his



FIG. 7.—SOUTH CAROLINA SEDIMENTARY KAOLIN MINE, SHOWING 15 FT. OF FIRST-GRADE CLAY AFTER STRIPPING 30 FT. OF OVERBURDEN.

visits to the field witnessed an apparently successful attempt at underground mining with close timbering. A short time afterward he was advised by the local manager that two of the miners almost lost their lives by a quicksand cave-in, and the tunnel had to be abandoned.

The majority of the plants have direct railroad connections, but some of them use narrow-gauge railroads for about 2 miles.

Three methods of stripping are employed at different mines. The old way, which is still practiced at most of the mines, is with pick and shovel and mule carts. Several have steam shovels, and one is equipped with a cable drag line and bucket. Advantages are claimed for each of these methods, but it would be hard to say which is the more economical.

In most cases the material has to be moved a considerable distance, so as not to interfere with future excavations. Rainy weather is never welcomed by the clay miner, as he not only has to suspend operations, but the rain washes iron-stained sand and silt over the face of his white clay bank. It also fills the pits with water, which has to be drained or pumped, and puts everything in such bad condition that it takes some time to put the mine in normal condition again. Generally, several grades of clay are produced from the same mine, and great care must be taken to see that the different grades are kept separate. Native negro labor is almost universally employed.

Only a few of the mines have washing plants, and these are necessary only where sand is present within the kaolin bed. The washed kaolin,



FIG. 8.—SOUTH CAROLINA SEDIMENTARY KAOLIN DRYING SHED, SHOWING KAOLIN READY FOR SHIPMENT IN HOGSHEADS AND IN BAGS.

however, is so thoroughly mixed that it makes a more uniform product and for that reason commands a higher price. One of the companies has installed a pulverizing plant through which the kaolin is passed after drying. This is claimed to mix the clay from different portions of the bed thoroughly, and gives a more uniform product to the consumer. Most of the beds being worked are so free from sand that the clay does not require washing, but is simply dried before shipping. As the climate is warm, open sheds with tiers of horizontal racks, upon which the lumps of clay are laid to dry, are in common use (Fig. 8). This, of course, is very economical and requires only one handling of the clay from the

pits to the sheds and rehandling when shipping. Several sheds are maintained by most of the companies, as one can be filled while another is being emptied, and it is always well to keep a reserve supply of dried kaolin on hand for emergency orders. Steam driers are used by the companies that maintain washing plants, but this is probably because they have exhaust steam which can be used to advantage.

Kaolin is shipped loose in box cars, in bags, or in hogsheads, according to the desire of the consumer.

Analyses of South Carolina and Georgia Sedimentary Kaolins

(South Carolina Clays by Hardin and Georgia Clays by Everhart)

	South Carolina			Georgia		
	1	2	3	4	5	6
SiO ₂	45.02	44.23	44.51	44.76	44.67	44.99
Al ₂ O ₃	38.98	38.92	38.12	38.41	38.76	38.59
Fe ₂ O ₃	0.77	2.31	1.75	0.63	0.85	2.11
TiO ₂	0.85	1.21	1.11	1.37	1.37	1.04
CaO.....	0.03	0.12	0.06	0.20	Trace	Trace
MgO.....	0.07	Trace	Trace	0.09	0.08	0.05
Na ₂ O.....	0.55	0.26	0.41	0.09	Trace	0.24
K ₂ O.....	0.26	0.30	0.32	0.35	Trace	0.11
H ₂ O.....	13.58	12.90	13.45	14.68	14.86	12.97
Total.....	100.11	100.25	99.73	100.58	100.59	100.25

III. PLASTIC KAOLINS OR BALL CLAYS

Florida Ball Clays

Distribution.—The white-burning clays of Florida which have thus far proved to be of economic importance are commonly known as the Florida ball clays. Exposures of these clays are confined to a belt extending from Putnam County south through Marion, Lake, Orange, and Polk counties. Mines have been operated only in the vicinity of Edgar, Putnam County, and in the vicinity of Richmond and Yalaha, Lake County. Other deposits of interest are reported in the vicinity of McMeekin, Putnam County; Bartow Junction, Polk County, and along Palatka River in Lake County.

Geologic Relations.—The area underlain with plastic kaolins occupies a portion of the Central Peninsular section of the State. Geologically this area is of Tertiary age and belongs to the Apalachicola group of the

Oligocene series. The Apalachicola group consists of a series of interbedded sands, clays, marls, fullers earth, and limestone. These sediments rest directly upon the Ocala limestone and in the Central Peninsular portion of the State dip gently toward the Atlantic Ocean.

Origin and Occurrence.—The kaolin-bearing beds are covered with an overburden of coarse quartz pebbles and reddish sands which reach a thickness of 10 ft. or more. The kaolin occurs in beds of coarse grayish-white sand which reaches in places a thickness of more than 30 ft. Cross-bedding is distinct throughout the entire thickness of the bed and particularly near its upper limits. According to Sellards,² "A well put down by the Edgar Plastic Kaolin Co. is reported to have passed through



FIG. 9.—FLORIDA BALL CLAY MINE, SHOWING REMOVAL OF KAOLIN-BEARING SANDS WITH FLOATING DREDGE.

coarse superficial sand 10 ft.; kaolin-bearing sand 30 or more feet; sticky blue clay with fullers earth beneath about 40 ft.; scarcely indurated shell stratum 20 ft.; the well terminated on a hard limestone rock at the depth of 90 ft."

Small particles of mica which are visible to the unaided eye are sparsely disseminated through the kaolin. This is evidence that the kaolin primarily had its origin in the residual clays of granitic rocks. It is very probable, however, that this kaolin was first deposited in Cretaceous times, and later eroded and transported to its present position. As this portion of Florida is several hundred miles farther from the crystalline area than is the Cretaceous horizon, it is reasonable to suppose that the particles of kaolin held in suspension for so great a distance

² Sellards, E. H.: *Second Annual Report, Florida Geological Survey*, p. 243 (1909).

would be more finely divided than those which were deposited in Cretaceous beds. This to some degree may account for the fact that the Florida clays are more plastic than the Cretaceous clays.

Mining and Preparation.—Unlike the North Carolina residual clays and the Georgia sedimentary clays, the Florida clays are mined by dredging (Fig. 9). As the country is very flat, and any open-cut work necessarily makes a depression in which water soon accumulates, dredging is naturally the most economic method of winning the clay. However, it is probably due to the fact that the beds are about 70 per cent. sand that dredging is successful. If these clays were as compact and stiff as the Georgia clays, they would not soak up well or yield to a suction pump. The recoverable kaolin rarely represents more than 30 per cent. of the material washed, and is reported to average not more than 20 per cent. After the overburden is removed the kaolin-bearing sand is lifted by the dredge and forced to a sufficient elevation above the washing plant. From this point it flows by gravity through a series of settling tanks, by which the sand and particles of mica are separated from the clay. From the settling tanks the kaolin slip passes to the filter presses and to the drying shed in the usual way.

Analyses of Florida Ball Clays

(No. 1, washed clay from Edgar; C. Langenbeck, analyst. No. 2, washed clay from Palatlahaha River; H. Ries, analyst)

	1	2
Silica (SiO_2).....	45.39	46.11
Alumina (Al_2O_3).....	39.19	39.50
Ferric oxide (Fe_2O_3).....	0.45	0.35
Lime (CaO).....	0.51	
Magnesia (MgO).....	0.29	0.13
Alkalies ($\text{Na}_2\text{O}, \text{K}_2\text{O}$).....	0.83	
Water (H_2O).....	14.01	13.78
Sulphur trioxide (SO_3).....		0.07
Total.....	100.67	99.94

Tennessee Ball Clays

Distribution.—The mining of Tennessee ball clay is confined to a narrow strip of land extending across the western portion of the State in a general direction which is about 20° east of north. This belt, beginning at the north, passes through Henry, Carroll, Henderson, Madison, Chester, and Hardeman counties. Henry, Madison, and Carroll counties have been the most important producers of ball clay up to this time. Mining for shipment to potteries has been most active in the vicinity

of Whitlock, Paris, and Henry, Henry County; McKenzie, Hico, and Hollow Rock, Carroll County, and near Pinson in Madison County.

Geologic Relations and Origin.—Probably the only clays in western Tennessee which might be termed white-burning clays are those plastic clays which are consumed by some of the white-ware pottery plants in their general mix. The best grades of this variety of clay are reported to burn to a cream white. As these clays occur associated with several other varieties of clay in the same geologic horizon and often in the same lense-shaped deposits, it is difficult to differentiate and define them. The other varieties, which can hardly be classed as white-burning, are stoneware clay, sagger clay, fire clay, and wad.

The most important clays belong to the LaGrange series of the Tertiary, although some occur in the Ripley sands of the Cretaceous. In geologic position they are very nearly the same as the Cretaceous clays of South Carolina and Georgia, only the Cretaceous beds in this area rest upon Paleozoic strata instead of upon the older crystalline rocks. The workable clay deposits of Tennessee generally cover but a few acres and are not nearly so continuous and uniform in character as those of South Carolina and Georgia. They are overlain with reddish sands and sand-clay beds, which vary in thickness from a few inches up to 30 ft. and more, according to the amount of erosion.

Though the Tennessee ball clays are somewhat different in chemical composition and physical properties from the South Carolina and Georgia paper clays, it is possible that they may to some degree have the same origin. Nelson³ attributes the primary origin of the clay substance embodied in these deposits to the disintegration of chert and the weathering of limestone in nearby Paleozoic rocks, but does not account for the presence of particles of white mica disseminated through the clay. Though this area seems far removed from the crystalline area of the Southern Appalachian region, it should be remembered that the waters of the Tennessee River, which must have once deposited their burden along this coast line, have their origin far into the crystalline area. Besides this, beds of white clay similar to those described above occur almost continuously in the Cretaceous from Tennessee through Mississippi and Alabama to Georgia. It would seem that a part of the clay may have had its origin in the residual clays of the crystalline rocks.

Mining and Preparation.—As the occurrence is very similar, practically the same methods of mining clay are practiced in Tennessee as in Georgia and South Carolina. The selling price, however, is not as high as that of the Georgia clays and therefore does not warrant washing or drying. It is shipped in box cars direct from the mines to the consumer.

³ Nelson, Wilbur A.: Clay Deposits of West Tennessee, *Bulletin No. 5, Tennessee State Geological Survey*.

Analyses of Three Different Grades of Tennessee Ball Clay

(Carl N. Zieme, analyst)

	1	2	3
Volatile.....	14.16	13.94	10.92
Silica.....	48.72	47.26	53.57
Alumina.....	32.89	35.85	32.82
Iron oxide.....	2.25	1.01	1.31
Calcium.....	0.60	0.58	0.62
Magnesium.....	0.65	0.68	0.38
Potassium.....	0.98	0.74	0.59
Sodium.....	0.14	0.45	0.21
Total.....	100.39	100.51	100.42

IV. MISCELLANEOUS CLAYS

Bauxitic Clay.—Associated with the bauxite deposits of Georgia, Alabama, and Tennessee are irregular deposits of clay, locally known as bauxitic clay. In many instances this material is very white and approaches kaolin in chemical composition. The clay occurs as horses in or irregular circular masses surrounding the deposits of bauxite. The contact with the bauxite may be sharp or the clay may grade into the bauxite. In the latter case the clay is always high in alumina. So far as is known, no serious attempt has ever been made to develop these clays commercially, but on account of their extent and physical properties they may yet prove to be of economic value.

Halloysite.—Halloysite is a clay-like substance which conforms closely to the chemical composition of kaolin, but has a considerably higher percentage of combined water. It is often waxy in appearance and usually has a greenish or buff color, sometimes white. Besides being associated with the bauxitic clays in small quantities, halloysite also occurs in pockety lenticular beds underlying the Fort Payne chert in Chatooga and Walker counties, Georgia. Other beds quite similar to those in Chatooga and Walker counties have recently been discovered about 8 miles west of Cleveland, in Bradley County, Tennessee. Samples of this halloysite have been burned repeatedly by the pottery manufacturers, and in every instance the color and vitrifying properties of the clay are reported to be equal to those of the highest-grade china clays. The uncertainty of the supply has probably restricted the demand for this material.

An interesting blue-black clay was recently found by the writer in the vicinity of Anniston, Ala. This clay is being tried out by several companies at present. A preliminary test by the Bureau of Standards

at its clay-testing plant in Pittsburgh, Pa., is reported about as follows: The clay is highly plastic, it burns to a white color, which is equal to or superior to that of commercial ball clays, and inferior to that of commercial kaolins. The clay is not as highly vitrified at 1,200° C. as ball clay, but may have value as a substitute for a portion of the ball clay in the production of ceramic mixtures. To all appearances this clay is a Coastal Plain deposit, and the only shaft which has been put down passes through 30 ft. of the material. It is reported by some of the potteries that have made preliminary tests that it resembles the German ball clays in appearance and physical properties.

USES

Few mineral substances have a wider range of commercial application than white clay. By far the largest portion of the white clay mined is consumed in the manufacture of pottery or white-burned products, which includes white-ware and cream-colored ware; bone, delft, and belleck china ware; sanitary ware, porcelain electrical supplies, architectural terra cotta, floor and wall tiles, and miscellaneous articles. The paper mills are probably the next largest consumers of white clay. In addition to this a large amount of white clay is used as a filler in paints, plasters, pastes, putty and school crayons.

PRODUCTION

The latest and most accurate figures on the production of white-burning clays in the United States are those published by the U. S. Geological Survey in the non-metallic section of *Mineral Resources* for 1913. Extracts from these figures are tabulated below:

Production of White-Burning Clays in the United States in 1913

States	Kaolin		Paper Clay		Ball Clay		Total		Average Value per Ton
	Tons	Value	Tons	Value	Tons	Value	Tons	Value	
North Carolina.....	16,332	\$139,629					16,332	\$139,629	\$8.55
South Carolina.....			31,568	\$120,520			31,568	120,520	3.82
Georgia.....			69,740	299,110			69,740	299,110	4.28
Florida.....					35,620	\$143,650	35,620	143,650	4.03
Tennessee.....					29,258	85,500	29,258	85,500	2.92
Total.....	16,332	\$139,629	101,308	\$419,630	64,878	\$229,150	182,518	\$788,409	\$4.32
Other States..	12,502	\$95,828			2,256	\$8,522	14,758	\$104,350	\$7.07

* Including some from other States.

It will be seen that North Carolina, South Carolina, Georgia, Florida, and Tennessee produced 182,518 tons, valued at \$788,409, while other States produced only 14,758 tons, valued at \$104,350. The highest average price per ton at the mines was for North Carolina residual kaolins, which was \$8.55, while the lowest price was for Tennessee ball clays, which was \$2.92. The average price per ton for all white clays produced in the five States referred to was \$4.32.

ECONOMIC ASPECT

In 1913 the United States produced white-burned products which had a total value of \$31,443,450, and in the same year imported white-burned ware valued at \$9,340,890. The total export of white ware from the United States in 1913 was valued at \$149,281; thus showing that the total value of the white-burned pottery products entered for consumption in the United States in 1913 was \$40,635,059.

In 1913, the imports of kaolin or china clay into the United States amounted to 268,666 short tons, valued at \$1,623,993, with an average value of \$6.04 per ton. According to Parsons,⁴ "This kaolin displaced a like amount of domestic raw material which, if properly handled, has no superior." Against this, in the same year, only 197,276 short tons of kaolin and ball clay were produced in this country, which had a total value of \$893,259 and an average value of \$4.53 per ton. Thus it may be seen that our domestic production of white-burning clays is only equal to about two-thirds of our imports in quantity, and half of our imports in value.

In consideration of the wonderful reserves of high-grade clays in the territory described above, it would seem that the United States, instead of importing annually large amounts of clay, should produce enough to supply her own demands, and even be a competitor in foreign markets. There are several good reasons, however, for this great difference between imports and domestic production, but only reasons which are due to existing conditions, all of which can be easily overcome.

One of the principal objections made by American potters to domestic pottery clays is that the product is not uniform in character, or, in other words, has not been properly prepared and classified before shipping. The potter justly demands a uniform product under guarantee, since the slightest variation in his ceramic mixture may cause the entire plant to be shut down until the irregularity is discovered and eliminated. Such variations may be of either chemical or physical character and may result in a lack of sufficient plasticity, a change in color on burning, a change in

⁴ Parsons, Charles L.: Mining and Treatment of Feldspar and Kaolin in the Southern Appalachian Region, *Bulletin No. 53, U. S. Bureau of Mines* (1913).

temperature at point of vitrification, a change in shrinkage, a tendency to warp and crack, and a failure to take the glaze.

To meet the requirements of the potter, therefore, the U. S. Bureau of Mines has wisely advocated a central depot where the output of a number of mines may be properly mixed and graded under the supervision of a trained ceramic chemist. Such a plant, if successfully engineered, should practically eliminate the present practice of mining to-day and shipping to-morrow, and should greatly stimulate the mining of kaolin by smaller operators. The output should demand a higher price than the clay is now bringing, and should capture the trade of our domestic potteries.

Such a plant, or holding company, could be operated successfully not only in the North Carolina residual kaolin district, but also in the South Carolina and Georgia sedimentary kaolin districts.

Another reason why American white clays are not more extensively used is because the clay mines, for the most part, are so remote from the manufacturing centers. Practically none of the white clays now being mined in the Southern Appalachian States are consumed in these States, and the bulk of the product has to be shipped by rail, more than 400 miles in the case of North Carolina kaolins, more than 600 miles in the case of South Carolina and Georgia kaolins, and more than 700 miles in the case of Florida ball clays. If these clays were properly prepared, and could be sold under guarantee to the pottery plants, there is no doubt that the potter would get closer to the source of his raw materials.

Estimates submitted by a number of clay miners in the North Carolina field show that this material, under favorable conditions, can be produced at a cost ranging from \$2 to \$4 per ton, and has a selling price of from \$8 to \$9 per ton at the mines. In the South Carolina and Georgia field estimates were submitted showing a range in cost of production of from \$1.50 to \$3 per ton with a selling price of from \$4 to \$6 per ton. The writer was not able to obtain figures on the cost of production of Florida ball clay, but, considering the great thickness of the clay-bearing beds and the economical methods of mining, it would be reasonable to assume that the margins of profit should compare favorably with those of the North Carolina and Georgia mines.

In view of the large amount of white clay consumed by American manufacturers, and the growing demand for domestic clays, it would seem that the clay-mining business in this country is almost in its infancy. Also, a review of the distribution of the clay resources in the United States would point to the fact that America must in the future look to the South for her supply of domestic white-burning clays.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Safety Methods and Organization of United States Coal & Coke Co.

BY HOWARD N. RAVENSON, GARY, W. VA.

(New York Meeting, February, 1915)

THE mines of the United States Coal & Coke Co. are located in the Pocahontas coal field, in McDowell County, West Virginia. Twelve plants have been opened and equipped, of which, by reason of the present business depression, only nine are now in operation. These twelve plants serve eighteen mines; some of the plants having several openings, in either the same seam or in different seams, tributary to the same tippie. Only nine of these mines are now being worked at the plants in operation.

Construction work was begun in 1902, and coal from the first three plants started was shipped in December of that year. An additional plant was opened in 1903, two in 1904, two in 1905, two in 1907 and the last two in 1908. On Jan. 1, 1915, the plants have been in operation an average period of 8.8 years.

The rules of the company, which have been in operation since work was first begun, are similar to those in use by the H. C. Frick Coke Co. at its mines in Pennsylvania, in which "Safety First" was incorporated as early as 1907; and it has always been the policy of both companies to look carefully after the safety of their employees. Although the field is an extension of the old Pocahontas field, the mines, located upon a new branch railroad, are away by themselves, and, as is usual in such cases, the class of labor attracted at first was temporary in character. The rapid development of the mines required the employment of many new men and officials each year; and, while the accident record from 1904 to 1909 was as good as the average of the field, it was felt to be not as good as it could be made, and a determined campaign was inaugurated for preventing accidents of all kinds. Fig. 1 shows the tonnage of coal produced per fatality, both total and for inside accidents only, at our mines; the tonnage per fatality of all West Virginia mines, and of those in the Pocahontas field.

The purpose of this paper is not one of congratulation on the results so far accomplished, since the operating officials are not at all satisfied that the maximum is in sight, to say nothing of its having been reached; but to describe the organization and methods that have produced these

results, with the hope that this information may be of service to others interested in accident prevention. Fortunately, the widespread safety movement which has been agitating the entire country for the past six years has awakened interest in this subject; and it is a very unprogressive company that is not now doing all in its power to safeguard its employees in every way. The time has now arrived when as much attention should be paid to serious accidents—those causing a loss of time of 35 days or more or a permanent injury—as to fatal accidents. In many

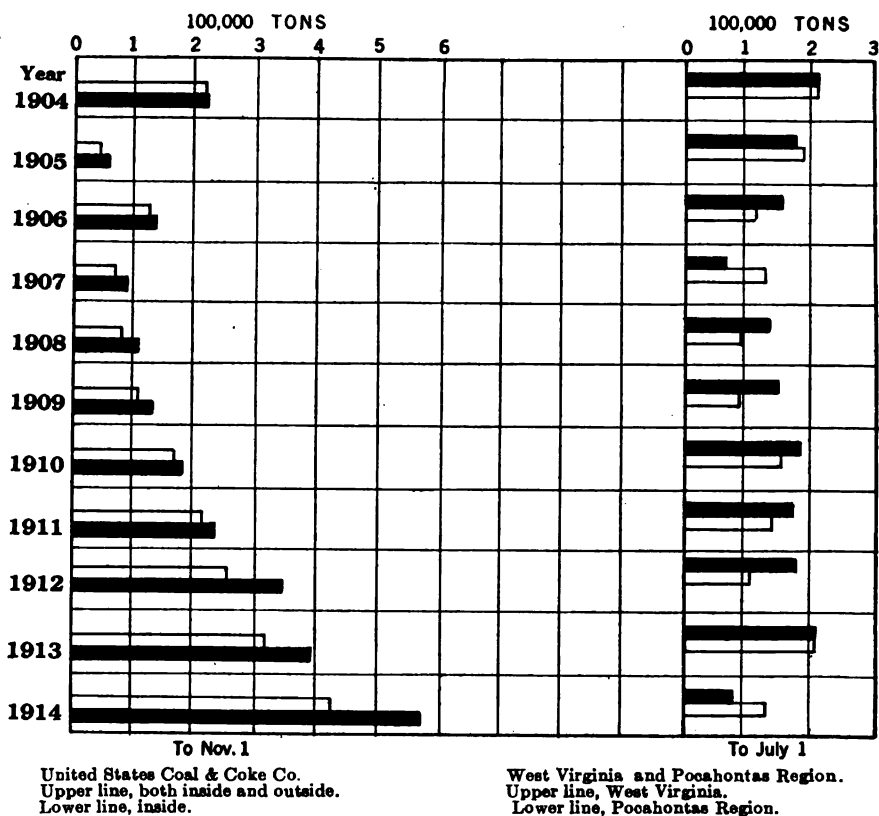


FIG. 1.—TONNAGE PRODUCED PER FATALITY IN THE MINES OF THE UNITED STATES COAL & COKE CO. AND IN WEST VIRGINIA AND THE POCAHONTAS REGION.

cases, the cause producing a serious accident would have produced a fatal one, with a few inches or a few seconds leeway; the difference being mainly a question of luck.

To those who have investigated mine accidents, the usual statement of the boss in charge of the mine that "he had been in the place an hour or so before the accident happened and had told the victim to set a post, but he waited to load another car before doing it," is entirely familiar; and,

while this is undoubtedly often the case and no reflection is intended on the hard-worked mine foremen who make it, it reveals a condition which must be removed before success in accident prevention can be expected. In our opinion, the first essential to success in work of this sort is the recognition of the fact that it is the duty of the employer, as far as possible, and by every means in his power, to prevent injury to the employee. So long as the view prevails that when the employee has been notified of any dangerous condition the management's responsibility is ended, effective prevention of accidents cannot be achieved.

Early in 1909 our inside organization was increased by adding to the list of officials at each mine assistant mine foremen enough to provide one for each 25 men employed. To each assistant was assigned a definite territory, over which he had the same authority that the mine foreman exercised over the entire mine. Where the territory assigned to an assistant was very large, the limit to the number of men in his charge was fixed by the number of places that he could inspect easily in 3 hr. The average number of men under each assistant foreman is about 22.

The fundamental idea in appointing these men was to provide enough executive officials at each mine to allow each one to stay in any working place where any dangerous condition was found until that condition was removed; and this is the one thing rigidly required of these men. Since this system was started, several accidents have happened, because this instruction was not strictly carried out; and it is now believed, with the experience of practically five years, that if this rule is rigidly followed, and the remaining rules of the company complied with, at least three-fourths of the accidents now happening can be prevented.

To acquaint these men thoroughly with the instructions of our higher officials and with the policy of the company in its mine work, complete projections are made, showing the proposed workings of each mine for a period of two or three years in advance; the station at which each place is turned and its course; the weight of track to be laid; the widths of the various places; the size of trolley wire to be used; the area on top of overcasts; whether overcasts and stoppings are to be permanent or temporary; and various other details, so that the future work to be done can be thoroughly understood by every official in the mine. In addition to this, a synopsis has been made in the form of a blue-print book of the instructions issued from time to time through circular letters by the General Superintendent. A copy of this book is given as Appendix A. It contains, under appropriate headings, in language that can readily be understood, the standards of the company for ventilation and inspection; explosives and shooting; track work, line sights, etc.; timbering, slate work, etc.; and general rules and standards for electric work. With these are bound standard plans for room timbering and track work, for various widths of places and depths of cuts, room switches, clearances and supports for trolley wires,

track bonding around switches, placing trolley frogs, etc. Each assistant mine foreman is furnished with a copy, and is required to familiarize himself with its contents.

After such additional supervision had been furnished for the inside workings of the company, a system of systematic timbering was introduced, under which every room in the mine is timbered, in accordance with the standard plan referred to above, unless the roof in the place is exceedingly bad and requires more than standard timbering. In other words, no matter how good the roof may be in any particular place, the standard amount of timber is required to be set there, and the only exception to this rule is that if bad conditions develop, additional timbers must be set to make the place safe. It was found that this was the only way in which the working places could be properly timbered. If the amount of timber to be set is left to the judgment of the miners or of the assistant mine foremen a loophole is provided for accidents and for excuses in case of accidents. The systematic timbering is designed to be sufficient for all except the very worst conditions that are found, and in these places additional precautions must be taken. This systematic timbering has been a large factor in the reduction of accidents.

It is an important duty of the assistant mine foreman to see that the timbers called for are set as soon as the coal is removed sufficiently to allow their placing, and not to allow the miners to wait until the day's loading is completed before setting any posts. The assistant mine foreman must see that all timbers that should be set in any place are so set before he leaves the place.

All shooting is done by the assistant mine foreman by electric battery; the miners drilling, loading, and tamping the holes with clay, which is furnished in each working place. The charge limit is 2 lb. of explosive per hole and only one hole is shot in any place at one time. None but permissible explosives are used, and all coal is undercut before being shot. About 85 per cent. of the cutting is done by machines; pick work is done only in room pillars, and in places where the weight renders machine cutting inadvisable or unnecessary. When the miner is ready for a shot, the assistant mine foreman comes to the place, sees that all "bug dust" has been loaded; that the place is clean; and the hole properly loaded; and that no coal dust has been used in the tamping. He then attaches the cable to the exploder wires and, after the miners and himself have retired to a safe place, fires the shot. After this, he inspects the face and roof, instructs the miners about pulling down loose coal or slate and setting posts, and *sees that this is done before he leaves*. Where pick mining is done, he sees that the place has been undercut before it is shot and that no shooting in solid coal is done.

On all haulage roads, and wherever more than one car is hauled at a time, 2.5 ft. clearance is provided on each side of the mine car. This

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

REPORT OF.....MINE 191..

MR. E. O'TOOLE

GENERAL SUPERINTENDENT

DEAR SIR:-

I visited.....Places, No. Men.....on.....Section
and found it as follows:

1. Quantity of Air in last Cross Cut.....
 Timbering (State condition and numbers of places).....
 Clay Ballast on Haulage Roads.....
 Frogs, Switches and Derails not protected.....
 Bad Slate in Rooms and Ribs.....
 Bad Slate on Headings.....
 Powder Caps.....Tamping for Shots.....
 Dangerous Practices with Stock.....
 Pieler and Safety Lamps.....Foreman's Cane.....
 Danger Signs.....Letters Posted Regarding Accidents.....
 Condition of Mine Cars.....
2. Dirty Coal.....
 Slate and Rash Piles.....
3. Cutting—Narrow.....Shallow.....
 Timber Drawn.....
 Condition of Roads.....
 Defective Wiring.....
 Line Sights.....Light Cars.....
 Condition of Live Stock.....
 Feed and Water given Stock.....
 Injured Stock in Stable from This Section.....
 Mining Machines out for Repair.....Safety Mottoes in Place.....
 Foremen's Record Book.....Material covered up or lost.....
 Condition of Pumps and Motors.....
- Miscellaneous.....

(When anything is wrong, give number of place, etc.)

MERITS?.....

Inspector

covers all places in the mine except rooms. On all haulage roads, at intervals of not more than 175 ft., are placed electric lights, which not only allow the trolley wire to be plainly seen but have led to a marked improvement in the quality of the haulage track.

Trolley wires are protected by hanging boards at all places where men or mules cross under them. The current used is of 275 volts.

All officials—Superintendent, Mining Engineer, Mine Inspector, foremen and their assistants—are instructed not to leave any place in which any dangerous condition is found until that condition has been removed.

All new men employed inside are instructed in their duties by the Mine Inspector, and this fact, with their names, is noted on his report. The Mine Inspector and Mining Engineer also measure the amount of air in circulation and report the quantity passing the last break-through of each pair of headings; 12,000 cu. ft. per minute is the amount required by the regulations. The form used by the Inspector is shown on the preceding page.

Both inside and outside, the careful guarding of machinery was inaugurated. After an experience of two or three years and the injury of several men in machines thought to be well guarded, it was found better to design the guards so that it would be difficult to get at the machinery, even to oil it, rather than to make this work easy and run the possible chance of an accident. In other words, it was found necessary to make access to all machinery a matter of some trouble, even to those maintaining it.

The class of men available for assistant mine foremen was not at first, as good as was desired. The problem, as in all accident-prevention work, was one of education; and it was found somewhat difficult to convince some of the men, particularly the older ones, that the company actually meant "safety" to be the first consideration. In fact, at the annual meeting held more than a year after this policy was first insisted upon one of our oldest mine foremen declared that he really did then for the first time believe that the company meant what it said in this matter.

In order to stimulate the interest of the assistant foremen in their work and to make the prevention of accidents a personal matter with them, a premium system was started in May, 1910, and has been, with some modifications found necessary, in effect since that time. Under this system, an assistant mine foreman having a clear accident record for a month receives a bonus of \$5; should his record be free from accidents for six consecutive months, he receives a special premium of \$10 per month in addition to the \$5 already mentioned; and this bonus of \$15 per month is paid him as long as his record remains clear. Should he have an accident he is charged with demerits for each man who is injured under his charge each month at the rate of 10 demerits for each minor, 20 demerits for each serious and 40 demerits for each fatal accident. No person having 10 or more demerits to his discredit at the end of each month is entitled

to any premium, but if he has less than 10 demerits he will receive the same premium as before. In addition to the accident record, no person is eligible for a premium for any month in any position who has not worked in that position every working day but one during the month, unless he shall have been promoted during the month from one position to another and is eligible in both positions. The intention of this requirement is to make the foremen work regularly. A number of accidents have occurred while the regular foreman was not working and his substitute was not equally familiar with the conditions. The work of every man must be satisfactory to his immediate superior. If it is not so, the superior has the right to charge him with demerits at the rate of 10 per month. This is a disciplinary measure, since it was found that a few foremen did not take sufficient interest in the prevention of accidents to attend the weekly meetings of the officials to discuss and investigate accidents which had occurred. But this provision has been used very seldom. Any assistant foreman in whose section the company's Mine Inspector finds any dangerous practices or dangerous conditions which might cause accidents, is charged with five demerits for each visit at which such conditions are found. This provision is required because an assistant mine foreman might permit dangerous conditions and practices on his section and still be fortunate enough not to have an accident. On the other hand, if no dangerous conditions or practices are found by the Inspector and everything is satisfactory, the assistant foreman is given a credit of five merits; and each assistant foreman who does not have an accident during any month is given a further credit of five merits, which goes toward reducing the number of demerits standing against him until all the demerits are canceled, after which no further merits are given until he has again received demerits. The men are not permitted to accumulate merits, as this would have a tendency to make them more or less careless after they had accumulated a large number. The mine foreman is charged with all the demerits received by the various assistant foremen under him, and is credited with all the merits received by them. In the case of mine foremen, the premium for a month's clear accident record is \$10 and the special premium for six continuous months' clear record is \$15. Thus a mine foreman having a clear accident record of longer than six months will receive a bonus of \$25 per month so long as his record remains clear, the assistant foremen under the same conditions receiving \$15 per month. This premium or bonus is not considered as a part of the wages but is strictly in the nature of a reward for faithful services rendered to the company in the prevention of accidents. The mine foreman's account is not charged with any demerits of the assistant foremen when these are given for the neglect of duty or causes other than accidents. If a foreman or assistant foreman leaves the employ of the company and later re-enters it he assumes all demerits standing against him when he left.

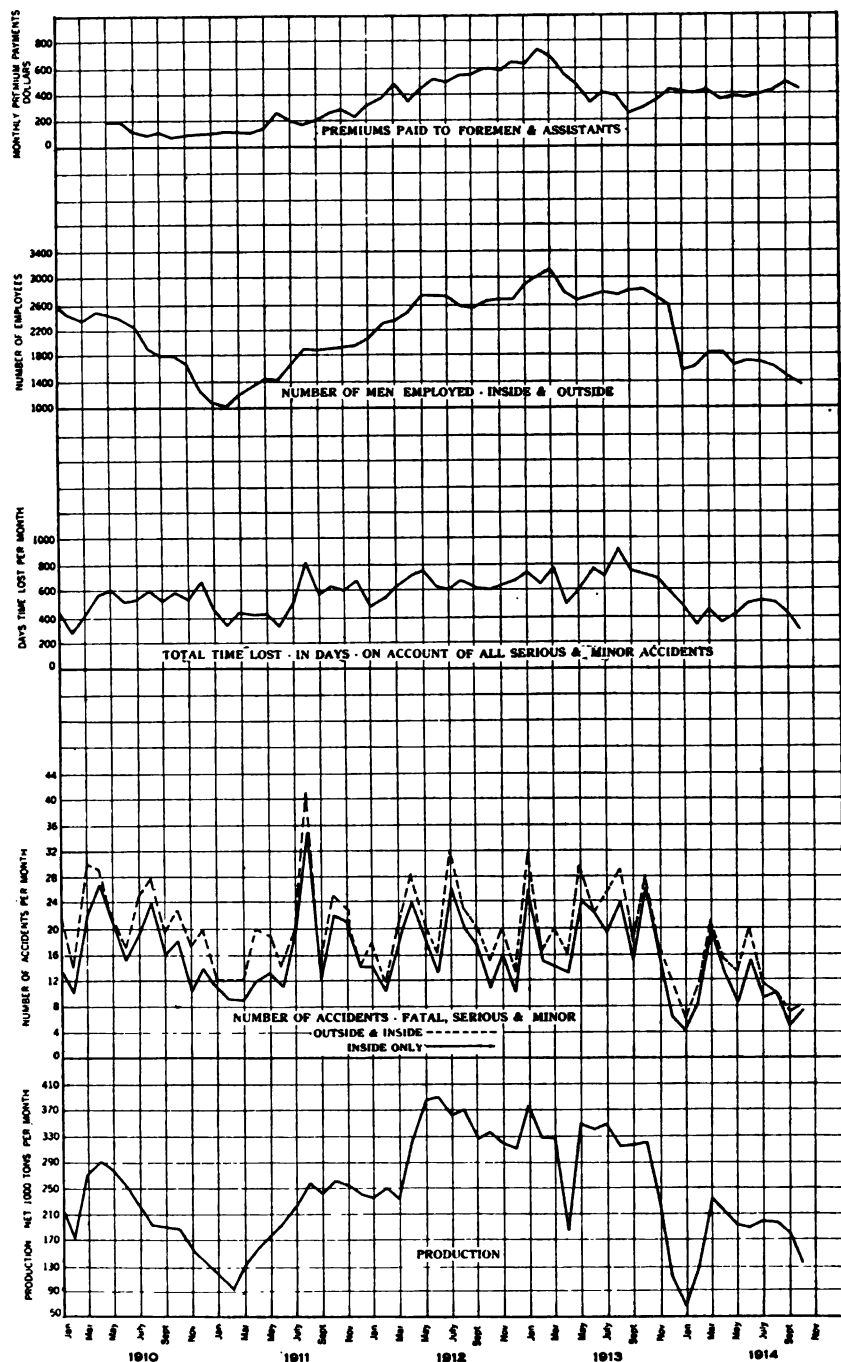


FIG. 2.—ACCIDENT AND PRODUCTION STATISTICS.

At the beginning of each month a statement is made showing the foreman and assistants at each mine, together with the number of demerits charged against them in the previous month, the number of demerits or merits received during the month, the number at the first of that month, the number of months' clear record, and the premium paid; and a copy of this statement is furnished each foreman and assistant foreman. This statement is looked forward to with a great deal of interest by the men and if any error is detected in it the matter is at once called to the attention of the person in charge of this work. The form of this statement is shown herewith.

United States Coal & Coke Company
Premium Statement for No Accidents—Month of September, 1914

Wks No.	Name	Occupation	Demerits		Merits Sept.	Demerits ‰	No. mos. clean record	Pre- mium	Spec. prem.	Remarks
			%	Sept.						
2	T. J. McParland	M. For		10	20		8	10.00	15.00	
	Mike Cassett....	Asst.			5		7	5.00	10.00	
	M. E. Maloney..	Asst.			5		6	5.00	10.00	
	J. P. Hanley.....	Asst.	5		5		4	5.00		
	I. C. Yates.....	Asst.			5		15	5.00	10.00	
	Jas. Barrett.....	Asst.	20	10		30				Not work full mo.

Since the adoption of the premium system in May, 1910, the company has paid 2,652 premiums, amounting to \$19,005, or a cost of 0.15c. per ton.

In Fig. 2 a series of curves is given showing, since January, 1910, the production; the number of accidents, including serious and minor, both inside and outside, and inside only; the total time lost in days per month on account of serious and minor accidents; the number of men employed, both inside and outside, and the premiums paid to the foremen and their assistants for each month.

From Fig. 1 it will be noted that since 1909, when the accident prevention work was really started and actively followed, the number of tons produced per fatality, both inside and outside, has risen from 107,323 tons to 428,962 tons (to Nov. 1, 1914), and the production per fatal accident inside only has risen from 128,788 tons to 571,949 tons. The systematic timbering and the fact that all headings and places on which more than one car in a trip passes have ample clearances on both sides have been important factors in the reduction of accidents.

At each mine a Safety Committee of three workmen is appointed for a period of three months, and at least once in that time these men make a complete inspection of the mine and report any places which they find in dangerous condition or make any recommendations they can toward the safety of the mines. These reports are forwarded to the General Superintendent. The form used is shown herewith.

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Report of SAFETY COMMITTEE No. _____ Works _____ 191 _____

NOTE: The Safety Committee will make their examination and recommendations beginning at the Coke cars on the coke yard and the cars on tippie sidings, through the tippie into the mine, including the coke yards, tippie, substation, haulage roads, traveling ways, hauling of the coal, mining, timbering, etc.; also including all machinery to see that it is properly guarded, or any other place in or about the works where men are working or machinery is located, including construction work.

REPORT OF CONDITIONS

(Do not crowd work. Use several sheets if required)

RECOMMENDATIONS

SAFETY COMMITTEE

Name

Occupation

Name

Occupation

Name

Occupation

Noted:

Assistant Foreman

Foreman

Superintendent

NOTE: This Report must be sent to the office of the General Superintendent at the first of each quarter of each year.

Each month the General Safety Committee, composed of the General Superintendent, Chief Engineer and Mine Inspector, meets and considers all fatal or serious accidents which have happened during the preceding month, or any other accidents having any unusual features, and have before them, if possible, the victim or any witnesses to the accident, the assistant foreman in charge of the section on which the accident occurred, the mine foreman and the Superintendent. The statements of these witnesses are heard, their evidence is weighed, and measures to prevent similar accidents are recommended. In cases of neglect on the part of the assistant foreman recommendations are made as to punishment. A stenographic report of these investigations is made for our files.

Shortly after the first of each year, beginning in 1910, a meeting has been held, at which are present the general officials of the company and all of the superintendents, mine foremen and their assistants, and all outside foremen; the purpose of the meeting being to bring the assistant foremen into closer personal contact with the general officials than can be done in the ordinary course of business. After the inner man has been provided for, speeches are made by various members detailed for that purpose, about the best methods of preventing accidents or similar subjects, and ex-members of the staff, State mine officials or other persons interested in the prevention of accidents are present and give short discussions on subjects of interest. These meetings have undoubtedly been of great service in stimulating and maintaining the interest in safety work, and we believe they do more to impress the assistant foremen with earnestness in this work than any other single thing.

In the case of any accident, a letter is prepared in the General Office, giving, in a short manner, the details of the accident and impressing upon the foremen and assistant foremen the lesson to be learned from this and the necessity of guarding against similar occurrences. A copy of this letter is sent to each mine and posted on the bulletin board near the pit mouth. In case of serious or fatal accidents, the Mining Engineer visits the place and, where necessary, a sketch is made showing the details of the working place, and prints of this sketch are also sent to each mine and posted on the bulletin boards. In case of fatal accidents, inspection is also made by the company's Mine Inspector, Chief Engineer, or General Superintendent.

At the beginning of each month a print is sent to each mine, on which is shown the number of fatal, serious, and minor accidents which have happened at each mine during the current year and the number that have happened during the month just preceding. This print is posted on the bulletin boards, informing every one at the mine of the number of accidents which have happened at all the mines and showing the relative standing of each one in this respect. Such a print is shown in Fig. 3.

At various times, when an accident of unusual nature or unusual in-

teret has occurred, a circular letter is sent to the assistant foremen to be posted in their sections, calling their attention to the matter and warning them against similar occurrences. In addition to this, safety signs,

SAFETY THE FIRST CONSIDERATION.											
UNITED STATES COAL & COKE CO.											
GARY, W. VA.											
ACCIDENTS FROM JAN 1 1914 TO NOV. 1, 1914 INCL. SHOWN IN RED. ○											
ACCIDENTS DURING OCTOBER SHOWN IN YELLOW ●											
NO. 1 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 2 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 3 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 4 MINE ACCIDENTS MINOR SERIOUS FATAL		
		○	○○○○ ○○○○ ○○○○ ○○○○ ○○○○		○	○○○○ ○○○○	○○		○○○○ ○○○○ ○○○○	○○	
NO. 12 MINE ACCIDENTS MINOR SERIOUS FATAL			○○○○ ○○○○ ○○○○ ○○○○								
			●●		●	●			●		
NO. 5 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 6 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 7 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 8 MINE ACCIDENTS MINOR SERIOUS FATAL		
○○○○ ○○○○ ○○	○○○	○	○○○○ ○○○○ ○○○○ ○○○○			○○○○ ○○			○		
			●			●●					
NO. 9 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 10 MINE ACCIDENTS MINOR SERIOUS FATAL			NO. 11 MINE ACCIDENTS MINOR SERIOUS FATAL			POWER PLANTS ACCIDENTS MINOR SERIOUS FATAL		
○○○○ ○○○○ ○○○○ ○○○○ ○○	○○○ ○○○ ○○○		○○○○ ○○			○○○○ ○○	○○	○	○○○		
●											

FIG. 3.—FORM OF ACCIDENT RECORD.

many of which are illuminated, are placed in the mines and around the plants, and are changed from time to time in order to stimulate interest.

Meetings of the assistant foremen, mine foreman, and Superintendent are held each week at each mine, at which the general details of their work are discussed, and particularly any accidents which have happened,

and the steps which should be taken in their particular places to prevent similar occurrences.

Last year, in co-operation with the United States Bureau of Mines, about 4,000 ft. of moving-picture film were taken, showing the complete details of the proper methods of mining. The object of these pictures was to show the miners exactly how the work should be done with safety and efficiency. These pictures have been exhibited at each of our different works a number of times; and it is our intention to repeat the exhibition from time to time in order that any new men coming in may be thoroughly posted. The class of men engaged in most mines, and, in fact, almost all men, can learn a great deal more through the eye by means of pictures of this sort than they can through printed or written instructions; and this film has been of great benefit in improving the efficiency and safety of our methods. Anything that tends to promote the efficiency of mining will undoubtedly, in the long run, promote its safety; and the safest mining will undoubtedly prove to be the most efficient.

Accident prevention is, as has been said, very largely a question of education; and our experience has been that the great majority of men are willing to take the precautions their experience, or knowledge, has taught them to be necessary, and that they appreciate the efforts being made by their employers to prevent accidents of all kinds. The great question is to reach the man who is actually doing the work at the working face with the superior knowledge and experience of the higher officials, and this can only be done by a system of detailed supervision and instruction through bosses having the necessary time to see that dangerous conditions are removed promptly. When the man at the face learns to carry out instinctively the directions given in the rules formulated for his benefit, the accident-rate will not exceed one-fourth of the present one.

Any work of this kind requires unceasing vigilance, careful supervision, and the never-ending stimulation of the spirit of care in all those having anything to do with it.

APPENDIX A

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE CO.

SYNOPSIS OF INSTRUCTIONS ISSUED IN CIRCULAR LETTERS FOR MINE FOREMEN AND ASSISTANTS

NOTE.—These instructions are intended to supplement the mine law, and the posted rules of the company. Should anything in them conflict with the mine law or the rules, the law and the rules must govern and be followed explicitly.

IN ALL WORK CONSIDER: FIRST, SAFETY. SECOND, QUALITY. THIRD, COST.

Approved, Sept. 25, 1911.

E. O'TOOLE, *General Superintendent.*

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Standards for Ventilation and Inspection

1. The last breakthru of every pair of headings, whether working or not, must have at least 12,000 cu. ft. of air per minute passing thru it.

2. On room headings, the air current must be properly checked with canvas curtains, to cause it to circulate to faces of rooms. Curtains must not be more than seven (7) rooms apart.

3. Ass't. foremen must examine each working place in their districts, and mark each place visited with date of month, before allowing men to enter them. Each idle place must also be examined each day, and a record of the examination must be made daily in the book provided for that purpose.

4. Before entering the mine, ass't. foreman must ascertain the condition of the ventilating apparatus from the fan man or the substation man.

5. When the fan has been stopped for one hour, or more, for any reason, all places must be thoroughly examined by ass't. foremen before allowing men to enter them, and a record of the examination must be made in the record book.

6. All permanent brattices must be built of incombustible material—concrete preferred.

7. Temporary brattices must be made of boards, well put together and with joints stopped up with cement.

8. If it is necessary to carry a brattice from the last breakthru to the face of any working place, use boards and not canvas.

9. Air measurements must be made weekly and be recorded in the book provided for that purpose.

10. For each shortage of air reported by monthly measurement of engineers, ten (10) demerits will be charged to the ass't. foreman in whose district it was found.

11. Ass't. foremen must carry Pieler testing lamps when making examinations, and all final tests for gas must be made with this lamp. When an ordinary safety lamp will not detect gas, try the Pieler lamp.

12. The presence of dust, as well as gas, must be noted in the record book.

13. Dust must not be allowed to accumulate in working places nor anywhere in the mine.

14. All dusty places on haulage roads must be sprinkled daily.

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Standards for Explosives and Shooting

31. No charge, of any explosive, in any one hole, shall exceed two (2) pounds.

32. Machine cuttings must be loaded in car before coal is shot down.

33. Before firing a shot, ass't. foreman must see that the place is free from gas and dust, properly posted and safe in all respects.

34. No one but ass't. foremen or persons designated by mine foreman are allowed to have possession of batteries or to fire any shot.

35. Before firing a shot, ass't. foreman must see that all workmen have withdrawn to a place of safety, and he must not be in the line of the shot himself.

36. Ass't. foremen must not fire any shot that is improperly drilled, drilled on the solid, or improperly tamped.

37. Ass't. foreman must not fire any shot within three hours after detecting gas in any place in his district.

38. Where more than one shot is to be fired in any place, one shot only shall be charged, tamped and fired, and an examination of the place must be made before firing the second or following shots. Only one shot shall be fired at a time. Ass't. foremen must carefully examine working places, after shooting, before allowing work to begin.

39. No explosives excepting flameless explosives, approved by the U. S. Bureau of Mines, shall be used.

40. All holes must be tamped with clay the full depth of the hole.

41. No shot that has missed fire shall be drilled out. Drill a new hole in such a manner that it will not come in contact with the explosive in original hole. Second hole must be charged lightly.

42. If shots miss fire, or are not fired for any reason, ass't. foremen must report the fact to the mine foreman, who must record the fact and the reason for it in his record book.

43. If necessary, water dust at working face before shooting.

44. Augers are to be made to gage, which must not exceed one and three-fourths (1¾) inches in diameter.

45. Use wooden sticks for tamping holes.

46. Shots must be fired by battery and not from machine or trolley wire.

47. Ass't. foremen must see that all caps or exploders are kept separate from explosives and at least thirty (30) ft. from them. Caps must be kept in a hole dug in solid coal.

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Standards for Trackwork, Line Sights, Etc.

61. Line sights are to be carried five and one-half (5½) feet from the rib in rooms and air courses, and in the centers of headings.

62. Sights in all places working must be extended each day.

63. Sight lines must be marked on roof by continuous chalk lines in all places.

64. All haulage roads on which more than one car is hauled per trip shall be at least five (5) feet high above the rail, and there shall be at least two and one-half (2½) feet clearance between any part of a car and the side of the heading at all places.

65. Permanent track must be kept within one hundred fifty (150) feet of face.

66. Track must be laid to line.

67. Where grades are given, track must be laid to grade.

68. All rash, slate, etc., is to be kept cleaned off haulage roads.

69. Ties are not to be over eighteen (18) inches centers on haulage roads.

70. Loaders must lay the track in their working places.

71. See that steel ties are used for room tracks and that all straight track is laid to exact gage, forty-eight (48) inches.

72. Permanent track must be bonded as it is laid.

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Standards for Timbering, Slate Work, Etc.

81. Where any dangerous slate is found, ass't. foreman must take it down at once before work is done near it, or anyone, or any trip, allowed to pass under it.

82. Loaders must post their working places. Proper arrangement and spacing of caps, posts, etc., is shown on standard plan attached, which must be strictly followed. Posts must be set in straight lines and be vertical. Cap pieces must be wedged tightly against roof, with wedge between post and cap.

83. Ass't. foremen must not O.K. nor allow any place to be cut where posts are more than six (6) feet from the face at the bottom.

84. All slate and loose rash must be taken down in haulage roads. Places must not be cut until slate or rash is taken down to face.

85. No permanent timbering will be allowed on haulage roads unless under special instructions from General Superintendent.

86. Slate or loose rash in manways is to be taken down, same as in haulage ways.

87. Airways are to be timbered in same manner as rooms.

88. Loaders must clean up slate falls in their working places, and must be paid extra for this.

89. Mining machines should start to cut on track side of place.

SAFETY THE FIRST CONSIDERATION

UNITED STATES COAL & COKE COMPANY

Standards for Electric Work

101. The general arrangement of hangers, etc., for trolley wire is shown on accompanying plans, which must be strictly followed.

102. Trolley wire must be hung six (6) inches outside of rail and must be as nearly parallel to it, both horizontally and vertically, as it can be.

103. Where roof is high, trolley hangers can be spaced thirty-three (33) feet apart on straight track. Where clearance between roof and wire is low, hangers can be spaced twenty-five (25) feet to twenty (20) feet apart on straight track; the wire must always be so hung that it cannot be forced against the roof by the trolley wheel passing under it. On curves, hangers are to be spaced as shown on standard plan, depending upon the radius of curve. Hangers must be close enough to prevent being bent sidewise by strain of wire. Trolley wire must be pulled tight and be dead ended with an insulated turnbuckle to an anchor bolt in roof or to a heavy post.

104. Where trolley wires or feed lines cross a heading or at turn out for side tracks, each must be protected by hanging boards, so that neither man nor mule can come in contact with wire.

105. Section line switches and sectional insulators must be installed on all trolley wire branches. Section line switches must be installed on machine and pump lines. Cable for wiring switches and insulators must be of same capacity as circuit controlled by switch. Ends of cables must be soldered solid and filed to fit terminal, or be soldered solid into feeder ear or terminal of switches and sectional insulators. Cables must be kept clear of switch boxes and porcelain tubes be used to bush holes, or holes have one-half ($\frac{1}{2}$) inch clearance all around cables.

106. Trolley frogs must be used at all turn outs and each frog must have an electric lamp beside it. Frogs must be located and supported by extra hangers, as shown on standard plan.

107. A clear space of at least two (2) inches must be allowed above trolley wire where it passes through a door, and hangers must be placed close to each side of door to prevent wire being forced up against wood.

108. Joints on trolley wire must be made with sleeves.

109. Feeder cables must be supported at intervals of twenty (20) feet on barn

hangers and special "Gem" insulators. Cables are to be placed twelve (12) inches outside of trolley lines and must be connected to them at intervals of two hundred (200) feet by feeder cars and solderless cable taps. Cables must be properly dead ended with cable clamps and insulated turnbuckles.

110. Machine wire must be supported on insulators, which must not be more than thirty-three (33) feet apart, and close enough to prevent wires touching posts, rib or roof. Ground wires must be put up in same manner as live wires. Where roof is good and not less than six and one-half ($6\frac{1}{4}$) feet above top of rail, wires should be put on roof at side of heading, and be eighteen (18) inches apart, the live wire being placed as close to rib as possible; where necessary to place wires on ribs and posts they must be twenty-four (24) inches apart. All wires must be dead ended on porcelain insulators. Porcelain tubes must be used around wires passing thru doors or curtains, and insulators must be placed close to each end of tubes. Joints must be well twisted together, with six (6) turns on each side. No hook joints will be allowed. Machine wires must be placed on same side of track as trolley wire. All wires must be pulled tight and be well tied to approved insulators. Machine wire should not be used in rooms less than four hundred (400) feet long. Where rooms are longer than this, room wires must not be connected to heading wires, excepting when machine is in operation in room. Each machine must carry one pair of jumpers to reach from heading to room wires. Insulators must be put on pins, and no wedging of insulators between wires or between wire and roof or coal or wood will be permitted. One wire only must be on one insulator.

111. All tracks on headings where electric haulage is used must be bonded when laid. Holes for bonds must be clean and bright and bond terminals be forced firmly against the metal. Splice bars must be left off joints until bonding is inspected by mine foreman or superintendent. All switches on haulage roads must be bonded as shown on standard plan attached, and all track should be cross bonded at intervals not exceeding five hundred (500) feet.

112. All light wiring must be thoroughly insulated from roof, rib or timbers, and must be tied to porcelain insulators with non-conducting material. Wire from lamp to ground must be on insulators, and wire from rib to rail be buried. No nails or staples will be allowed for fastening wire.

113. All insulated cables and wires must be kept free from grounds, same as bare wires.

114. Pipe lines must not be used solely for pump motor returns, but when running close to and parallel with a bonded rail, may be connected to rail at intervals not exceeding five hundred (500) feet.

115. Where lights are to be installed in mines, or damp, wet or dusty buildings, where waterproof sockets must be used and where lamp is not to be carried about, wires to socket must be soldered direct to the circuit wires, but supported independently of them. Porcelain cleats or split knobs can be used to support droplight. See plan.

116. Where lights are to be installed that are used for portable purposes, or where lights come in contact with surrounding objects, portable cord—*not lamp cord*—must be used, wires to be soldered direct to circuit wires and be supported independently of them with porcelain cleats. See plan.

117. Lamp cord will not be approved in wet or dusty places. Cord must hang free from rosette. No looping of long cords and no suspending from hooks or nails in ceilings will be permitted.

SAFETY THE FIRST CONSIDERATION
UNITED STATES COAL & COKE COMPANY

General Rules

121. Mines will begin work at 7 A. M. and stop dumping at 3:45 P. M.
122. All men must be checked in and out of the mine each day.
123. No one but employees are allowed to enter the mines unless accompanied by the mine foreman or some other official.
124. No person under eighteen (18) years of age shall be employed without first furnishing affidavit of age from his parents or guardian.
125. All work, excepting such repairs as cannot be done while operating or at night, must be suspended on Sundays.
126. Do not allow grease, oily waste or any other inflammable material to accumulate in pump houses, offices or anywhere else.
127. Do not allow wood, old boards, ties, posts or any inflammable material to accumulate in the mines.
128. Mining machine men must be in working places or at mine office at 4 P. M. to receive instructions from ass't foremen.
129. Unless otherwise instructed, one man only will be allowed per working place.
130. No machinery of any kind must be allowed to operate unless all gears and dangerous portions are fully guarded.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Development of the Butchart Riffle System at Morenci

BY DAVID COLE, MORENCI, ARIZ.

(New York Meeting, February, 1915)

THE appearance of the Wilfley table in 1897 marked an epoch in the art of concentration of ores. The table has merited and received an almost unprecedented measure of public approval, lasting through its whole patent life. It has been very little improved in itself, or improved upon by competitors. The new machines bidding for popular favor have been of the Wilfley family, but without ability to do markedly better work than the parent machine, or to do more work except by the use of superimposed decks, the latter having obvious disadvantages.

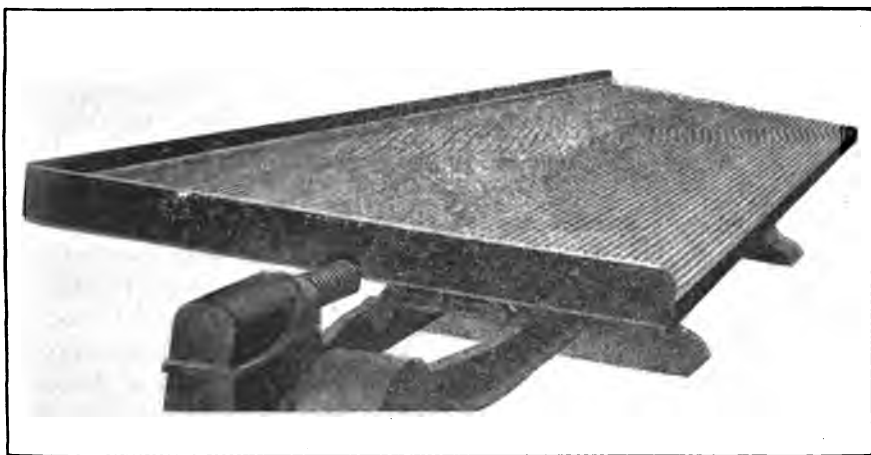


FIG. 1.—THE BUTCHART RIFFLE SYSTEM.

During the past three years there has been developed at Morenci a new type or arrangement of riffles applicable to the Wilfley type of concentrating table which corrects many of the objectionable features or limitations of the older system and obviates most of the difficulties encountered with it, particularly when handling ores having a low ratio of concentration. The new type is known as the Butchart¹ riffle system, and its general arrangement is shown in Figs. 1 and 2.

The old system accomplished stratification satisfactorily but has,

¹ Also known as the National table.

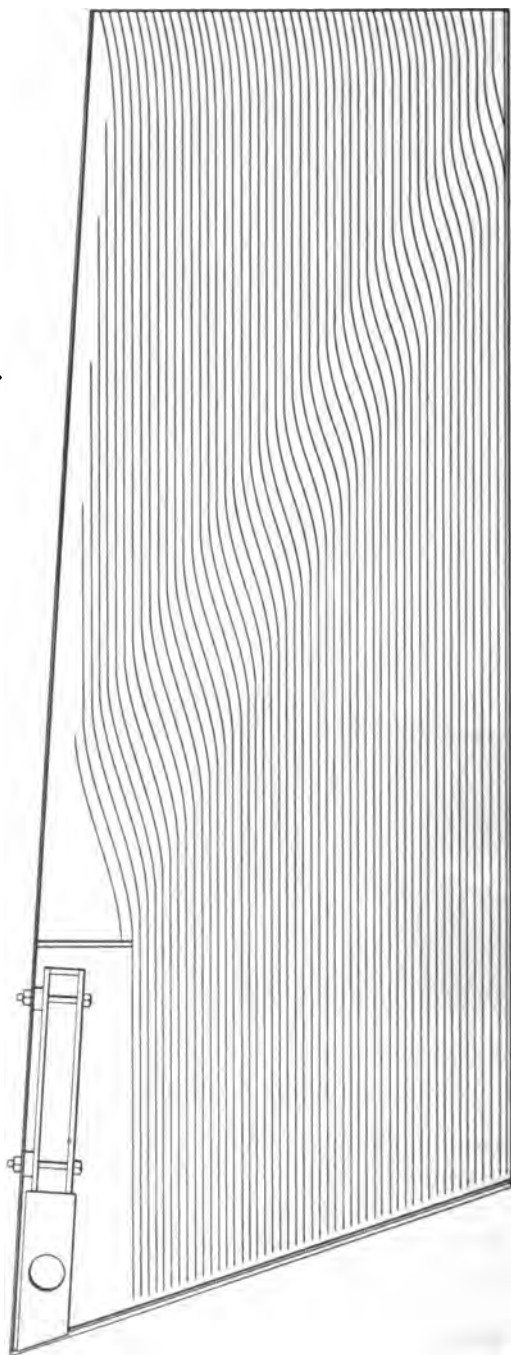


FIG. 2.—THE BUTCHART RIFFLE SYSTEM

insufficient means for separating the strata. The new system not only stratifies the minerals but is provided with ample means for segregating and separating them. The old system has a single cleaning channel, or zone, consisting of the diagonal depression adjacent to and parallel with the successively advancing ends of the riffles, and this is easily overloaded in a way to defeat in large measure the object desired. The new system has as many individual cleaning channels as there are riffles upon the table, and these are not easily overloaded because they are filled in succession and back each other up.

The old system is not adapted to the handling of very coarse table feeds, stopping for good work at about $2\frac{1}{2}$ mm. size, while the new system handles very large tonnages of what have hitherto been jig sizes, with a much simpler arrangement of machinery and with greatly improved general results.

The new system eliminates the necessity of hydraulic classification beyond the desliming stage. Desliming or classification of any kind is not required on coarse feed because the table rejection is usually dewatered and reground for future treatment, and when either primary or reground material is to be handled it is necessary only to remove thoroughly the slime either in spitzslutte or drag classifiers. When classification is carried further than this too much of the fine sand is eliminated, with detrimental effect upon the work of the new table.

The new system requires less water than the old. The water consumption of the new system is from 250 to 275 gal. per ton of feed; therefore, when these tables displace jigs the water consumption is reduced from 50 to 75 per cent., and when used instead of the old system it is reduced approximately 50 per cent.

With these advantages the new system is very important in wet concentration, particularly where the ratio of concentration to be practiced is medium or low, and when used in conjunction with a successful flotation treatment of colloid overflows it makes possible the construction of a most desirable wet concentrating plant, adapted to large tonnages in relatively small space, and for a minimum capital expenditure per ton treated.

In developing the table at Morenci, it was first recognized that the Butchart system of riffles gave great stability to the operation of tables under varying conditions of feed and water. Suddenly increasing the quantity of feed would not interfere with the ability of the table to make clean concentrate, and it was not necessary continually to be making adjustments of the table slope in order to bring the concentrate to the proper cutting-out point. These differences were in great contrast to the old system and discounted the indifference plus carelessness of the average operative now found in large mills. This consideration alone was considered sufficient to justify the adoption of the new system

in the Arizona Copper Co.'s No. 6 Concentrator, where this system of riffing has since been evolved into its present form.

At first the riffles were thin and the ultimate capacity of the table was unknown. Discoveries in the matter of capacity came about rapidly through experimentation. For example, it was found desirable to reduce the insoluble material in Hancock jig concentrate, which was a mixed product ranging in size from $\frac{3}{8}$ in. down to 200 mesh, containing 15 per cent. copper and 30 per cent. insoluble. The standard No. 5 Wilfley table was first installed for this work, but it was found impossible with this system of riffing to handle successfully the range of sizes or the excessive and variable tonnages coming from the jigs. Butchart strips were then substituted for the Wilfley riffles. These strips were thicker than usual, purposely making deeper channels, and it was found that very little work of adaptation was required to get excellent results. Heavy tonnages could be treated with a minimum amount of material sent to the middlings; the table responded to great variations of load; the insoluble material was reduced to less than 15 per cent.; and the coarse concentrate, instead of working down and arranging itself in the rear of the concentrate band, as it does with the Wilfley riffle, was discharged out of the top riffle into the concentrate launder; and time samples showed that the table load averaged about 80 tons per day.

Experiments were then tried on coarse unclassified feed up to and including the undersize of 4-mesh screen affording square openings of $\frac{3}{16}$ in. size. This feed had a concentration ratio of 15 into 1, and it was found that as much as 200 tons per day could be handled successfully, making clean concentrate containing a range of sizes from $\frac{3}{16}$ in. to 100 mesh and finer.

The plant was being operated at full capacity during a period of reconstruction and it became necessary to remove the intermediate Hancock jigs and to crush the primary-jig rejections in one operation in 8-ft. Hardinge mills direct to 20-mesh size for treatment on sand tables. A paddle-wheel classifier, which was a fairly good dealimer, was in use following the Hardinge mills. Floor space was limited by construction operations and but few tables could be accommodated at the time, so a few of the new ones were fitted with somewhat thicker Butchart curved strips, and the relatively rich product of the paddle wheel was fed upon them to what appeared to be maximum capacity, as judged by the appearance of the concentrate and panning of the tailing. Time samples then brought out the surprising fact that the tables were often handling more than 100 tons each, which was three times the load of neighboring Butchart machines and five times the load of ordinary Wilfley riffles, and were making concentrate carrying but 17 per cent. insoluble, with averages as follows: Feed, 95.2 tons, 2.10 per cent. copper; concentrate,

17.70 per cent. insoluble, 15.83 per cent. copper; tail, 0.86 per cent. copper; ratio, 12.1 into 1; extraction, 62.4 per cent.

The subsequent adoption of this system in a neighboring mill has greatly simplified the operation in its primary stages and has reduced costs without detriment to final metallurgical results. In making the change 10 of the No. 5 type Wilfley tables were rearranged for use with the Butchart riffles, and eight of these handle the complete ore tonnage with astonishing results, making it possible for the company to so change and simplify its flow sheet as to dispense with the use of a large amount of machinery with its accompanying expense for repairs, power and operation. This is illustrated by the comparative flow sheets (Figs. 3 and 4), which show the equipment used before and after the change, in which the following list of machinery was discarded:

- 13 42 in. by 9 ft. trommels
- 4 one-compartment Harz jigs
- 8 two-compartment Harz jigs
- 12 three-compartment Harz jigs
- 1 two-compartment shovel-wheel classifier
- 14 12 by 12 ft. settling tanks
- 17 Wilfley tables
- 4 No. 2 Deister tables
- 5 No. 3 Diester tables
- 1 V-dewatering tank
- 1 elevator
- 1 hydraulic sizer
- 2 dewatering boxes
- 1 vanner

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For more than a year the eight primary tables referred to have been handling an average of 1,600 tons of feed per day, the feed consisting of the undersize of 7-mm. screens, going to the tables without desliming, classification, or other preparation, with the typical results—the average of a long period—which follow:

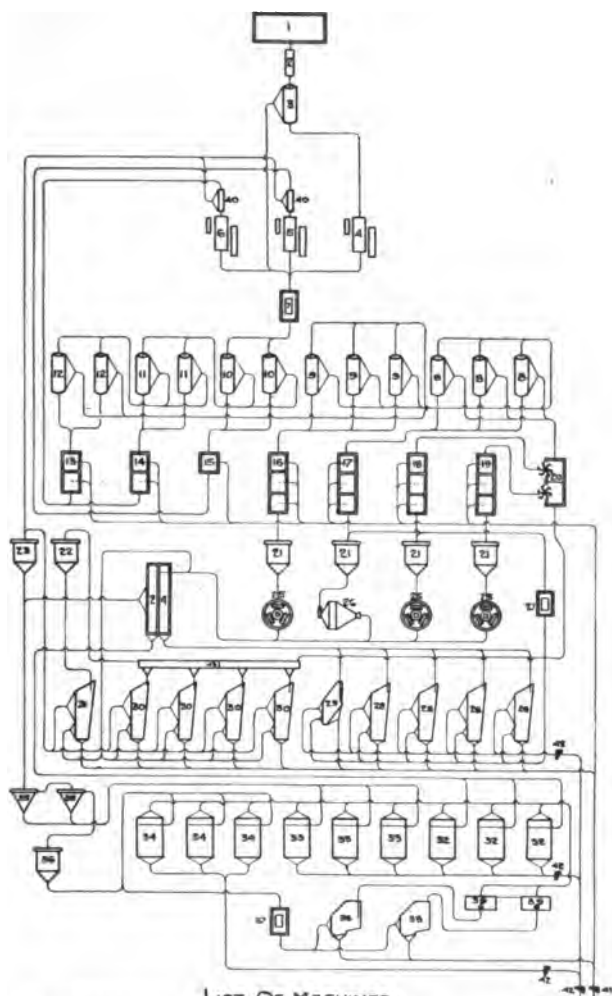
Tables with Butchart Riffles

Head		Concentrate			Reject, Per Cent. Cu	Per cent. Recovery	Ratio
Tons Each	Per Cent. Cu	Per Cent. Cu	Per Cent. Insoluble				
160	3.15	16.65	9.8	1.42	60.04	8.8:1	

Group of Harz Jigs (at same time)

35	3.19	17.43	11.9	1.63	53.95	10.1:1	
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The reject from the above was reground to $1\frac{1}{2}$ mm. size, deslimed by drag-belt classifier, sand treated on Butchart riffles and same sand



LIST OF MACHINES

NO.	DESCRIPTION	NO.	DESCRIPTION
1	750-TON HULL STORAGE BIN	22	4 12"X12" CONCRETE SETTLING TRAYS
2	1 PLUNGER-TYPE CRUSH PRESS	23	14 WOOD SETTLING TRAYS
3	1 42"X9' TROMMEL, 5/16" ϕ HOLE	24	1 2 COMP. DECK CLASSIFIER
4	1 SET 40"X16' TROMMEL ROLL TO 3/8"	25	3 8FT CHILSEN MILLS
5	1 SET 40"X16' TROMMEL ROLL TO 3/8"	26	1 36"X6' HARDING MILL
6	1 SET 40"X16' TROMMEL ROLL TO 7/16"	27	2 MIDDLE ELEVATOR
7	2 BUCKET ELEVATORS, 22" BELT 70' C/S	28	11 WILFLEY TABLES
8	3 42"X9' TROMMELS, 2 1/2" ϕ HOLE	29	4 N/2 DEISTER SLIME TABLES
9	3 42"X9' TROMMELS, 5 mm ϕ HOLE	30	4 WILFLEY TABLES
10	2 42"X9' TROMMELS, 5/16" ϕ HOLE	31	4 WILFLEY TABLES
11	2 42"X9' TROMMELS, 1/8" ϕ HOLE	32	10 FINE VIBRATORS
12	2 42"X9' TROMMELS, 7 mm ϕ HOLE	33	10 FINE VIBRATORS
13	4 2 COMP. HASTE JIGS	34	20 FINE VIBRATORS
14	4 2 COMP. HASTE JIGS	35	2 V-DEWATERING BOXES
15	4 1 COMP. HASTE JIGS	36	6 12"X12" SETTLING TRAYS
16	3 3 COMP. HASTE JIGS	37	2 TAILINGS ELEVATORS
17	4 3 COMP. HASTE JIGS	38	7 N/2 DEISTER SLIME TABLES
18	3 3 COMP. HASTE JIGS	39	2 DEWATERING BOXES
19	2 3 COMP. HASTE JIGS	40	2 DEWATERING SHAKING LAUNDER
20	1 2 COMP. SHOVEL WHEEL CLASSIFIER	41	1 HYDRAULIC STEEL 35 SPAGOTS
21	4 46"X56" DEWATERING TRAYS	42	4 AUTOMATIC SAMPLERS

TOTAL NUMBER OF MACHINES = 159

FIG. 4.—FLOW SHEET BEFORE ADOPTION OF BUTCHART RIFFLE SYSTEM.

simultaneously on standard Wilfley riffing, with the following comparative results of four days' continuous sampling:

One Table with Butchart Riffles

Tons	Head			Concentrate			Rejections			Ratio	Per Cent Recovery
	Per Cent.	Cu	Insol.	Per Cent.	Cu	Insol.	Midd.	Tail	Midd. and Tail		
							Per Cent.	Per Cent.	Per Cent.		
101	1.66	12.46	11.3	0.91	0.55	0.61	11.3 : 1	66.50			

Comparative saving and loss: Middling, 6.94 per cent.; concentrate, 66.5 per cent; tail, 26.56 per cent. Water consumption, 16 gal. per minute, 256 gal. per ton.

One Table with Wilfley Riffing

Tons	Head			Concentrate			Rejections			Ratio	Per Cent. Recovery
	Per Cent.	Cu	Insol.	Per Cent.	Cu	Insol.	Midd.	Tail	Midd. and Tail		
							Per Cent.	Per Cent.	Per Cent.		
35.5	1.47	13.21	10.9	2.26	0.59	0.76	17.5 : 1	51.25			

Comparative saving and loss: Middling, 14.75 per cent.; concentrate, 51.25 per cent.; tail, 34.0 per cent. Water consumption, 12 gal. per minute, 867 gal. per ton.

A table with Butchart Riffles was afterward tested against a group of five standard Wilfley tables, the latter handling as nearly as could be gauged the same tonnage as the one Butchart, with the following averages for four days' continuous sampling:

One Table with Butchart Riffles

Tons	Head			Concentrate			Rejections			Ratio	Per Cent. Recovery
	Per Cent.	Cu	Insol.	Per Cent.	Cu	Insol.	Midd.	Tail	Midd. and Tail		
							Per Cent.	Per Cent.	Per Cent.		
96.8	1.57	14.32	17.4	1.00	0.74	0.79	17.3 : 1	52.58			

Comparative saving and loss: Middling, 11.54 per cent.; concentrate, 52.58 per cent.; tail, 35.88 per cent.

Five Tables with Standard Wilfley Riffles

Tons to 5 T	Head			Concentrate			Rejections			Ratio	Per Cent. Recovery
	Per Cent.	Cu	Insol.	Per Cent.	Cu	Insol.	Midd.	Tail	Midd. and Tail		
							Per Cent.	Per Cent.	Per Cent.		
115.9	1.39	15.20	16.8	1.55	0.68	0.81	24.8 : 1	44.07			

Comparative saving and loss: Middling 15.99 per cent.; concentrate, 44.07 per cent.; tail, 39.94 per cent.

It will be noted in these tests that the insoluble is slightly higher in the Butchart concentrate. This is because this type of riffing cuts more oxidized copper into the concentrate than the Wilfley system does. The saving of this oxidized mineral at this stage is very advantageous, for it is as high in copper as milling can make it, but being lighter it goes off in the Wilfley system in the middling *and thus raises the copper content of the rejections and increases the ratio of concentration at the expense of the extraction.* It will be noted further that the Butchart riffle system has evidently removed all the free mineral from what would be the middling zone to the concentrate. Of course, the Wilfley middlings could be re-treated and the free mineral removed, but this complicates the operation and is unnecessary when the Butchart system is used. Middling on the Butchart system is a true middling, consisting of grains needing further crushing to free the mineral, together with the lighter part of the oxidized mineral, etc., and with a small amount of free but very fine mineral grains. *The middling on the Wilfley system consists of the above, plus what free and relatively coarse mineral the cleaning zone, on account of its overload, has failed to separate.*

In another plant and district, on a type of ores very different from those of Morenci (having no water-soluble oxidized or carbonate copper), loads as heavy as 200 tons per day of thoroughly deslimed but otherwise unclassified feed, being the undersize from $1\frac{1}{2}$ -mm. screens, assaying an average of 1.34 per cent. copper, have been treated with an extraction of 54 per cent. of the copper content, as compared with an extraction of 38 per cent. effected upon tables running in parallel out of the same launder but equipped with a modified Garfield system of riffing. The concentrate in each instance was cut to 20 per cent. insoluble, which is the smelter standard worked to in this case. The rejections of the Butchart table contained very little copper except in the form of attached mineral, while the other tables rejected much free mineral, through inability to clean and segregate it to the concentrate launder.

The modification in the Garfield system of riffing above referred to consisted of cutting off the ends of the riffles diagonally on the concentrate corner of the table to make a cleaning zone there, at the same time retaining the deep channels of the Garfield system.

If the Butchart riffles were to be straightened out, i.e., laid without curves, the result would be the Garfield system of riffing, and it will be realized that the table would then require more side slope to prevent the whole load from going over the concentrate end; and on account of the violent agitation produced when enough side pitch is given to the table to carry the sand into the tailing launder properly, the fine material will not have a good chance to settle and much more of it will go over into

the rejection than would be the case when the curves are used, for with the new system the side slope is such that a gentle rolling over of the load is accomplished. The movement of the load bodily along toward the concentrate end is checked by the curves and these enrich the product passing them to whatever extent the plan of operation requires. Thus the Butchart system makes a superior roughing as well as a superior finishing table. When simple roughing is the work to be done the curves are modified to suit that class of work.

In another district a company has installed a "pilot mill" to determine the best flow sheet adapted to its ores. It has developed in this mill that two Butchart tables handling the drag-belt deslimed product of Hardinge mills, assaying 1.35 per cent. copper, ratio 8.5 into 1, are making concentrate averaging 9 per cent. copper, 8 per cent. insoluble, 32 per cent. iron, and with middling plus tail that will average 0.35 per cent. copper. The extraction in this case is from 75 to 80 per cent. of the copper present.

In this pilot mill the roughing out of the concentrate of the first separation stage above the regrinding mill is done on a Butchart table on unclassified and not deslimed feed that has passed a 7-mm. screen, with the same generally good results as to production of finished concentrate that have been noted in the previous cases.

This pilot mill is proving that with a simple arrangement of crushing machinery reducing the ore to 7 mm. for the first separation by the new system, the rejection through regrinding mills for finished reduction, desliming of the reground sand, treatment of the sand over Butchart tables and of the lime by flotation, a very high percentage of the total value will be recovered. Most of the final rejections are made from Butchart tables and are sand tailing containing the minimum amount of copper noted above, indicating very cheap costs for construction of plant and, on account of its simplicity, very simple and inexpensive operation throughout.

Operation Analysis

In order to understand the marked change in table conditions brought about by this new system of riffing, it is necessary to analyze in some detail the method of operation in the old system and compare it with the new one.

In the work of each system the coarse and more cubical pieces are more free to move and more easily rolled by the carrying water than are the smaller and flatter particles, and their arrangement upon the table is in accordance with the law governing the angle of repose of particles partly suspended in water. The tilt sidewise given to the table causes the wash water to move more speedily and also provides more slope to assist the rolling-over action of the mass being treated, with the result

that the coarser particles of low specific gravity go clear across the bed of pulp upon the table and arrange themselves in the rear of the mass moving forward, while the successively smaller sizes, with those having successively greater specific gravity, tend to arrange themselves farther up on the slope and farther along toward the discharge end. Thus the forces at work tend to arrange the bed upon the table with successively smaller grains from rear to front and successively lighter grains from bottom to top. The differential motion moves the whole mass lengthwise, while gravity is rolling it crosswise the table deck, and the heavy bottom stratum of mineral consisting of medium-sized and fine particles appears first upon the cleaning zone of the Wilfley system, and goes farthest up the slope against the influence of the dressing water tending to wash it back.

Each channel of the Wilfley system properly stratifies its portion of the minerals and then becomes a conveyor to bring the mineral strata under the influence of the wash water in the single diagonal cleaning zone just beyond the ends of the riffles.

When the quantity of mineral of high specific gravity is in a relatively small proportion, the recoverable portion of it is pushed forward in the grooves by the differential movement beyond the terminals of the riffles into this zone, where it encounters the clear water on a new slope and where it is rolled over itself, while the fine sand is eliminated in the process of dressing, the finer and heavier particles finally occupying the thinnest edge nearest the wash-water box, the coarser working down and arranging themselves in the rear of the mineral band. This is the situation desired and the one that results in good separation, but this action cannot take place except where the layer of mineral is sufficiently small in volume so that it can all be cleaned outside and beyond the ends of the riffles; which means that if the separation is to be satisfactory the table must be fed with tonnages affording only the amount of concentrate which the limited cleaning zone can successfully handle, and that if larger tonnages of low-ratio material are handled the cleaning zone will fail proportionately with the overload, the clean-mineral stratum will not emerge from the riffles and will be covered by the middling stratum to a large extent, and therefore will go off as middling, or to the detriment of the concentrate if cut out with that class.

With the Butchart system the entire surface of the table is covered with riffles which are made to perform useful work. Each of the channels becomes a distinct separating device which cleans its own concentrate in the curve between the riffles instead of upon a smooth unriffled surface. It has the ability to perform this function on the upper one-half of the deck, on the large quantities of concentrate produced, the balance of the surface being free to treat the more difficult portion of the concentrate, to clean up middling, and to be ready to handle larger quantities of concentrate when the load or ratio fluctuates.

The riffles extend the full length of the table, are exactly alike as to dimensions, and taper from the rear, or mechanism, end to the front, or discharge, extremity. For average operations they usually taper from a height of $\frac{1}{2}$ in. at the rear to $\frac{1}{8}$ in. at the front end. These dimensions are varied to provide such concentrate-carrying capacity as may be demanded by the average ratio of concentration in the material to be treated. They may be, if required, 1 in. in the rear, tapering to $\frac{3}{8}$ in. at the tip, or discharge, end. The curved deflection of the riffles together with the transverse inclination of the table produces in the curved channels a downward slope toward the rear, or mechanism, end of the table, and the contents of the channels have to climb up this slope against a gentle stream of water passing down it. As the channels in the cleaning zone are not parallel to the direction of motion, the straight-line movement of the table causes a circular agitation or side shake to be imparted to the contents of this portion of the channel, subjecting all of the contents to a vanning action which causes the fine-silica content of the strata to pass down the channel with the clear wash water and to pass over at the lower end of the curve, and, notwithstanding a great variation in the concentrate sizes, effectually cleans the concentrate while climbing up the slope. At the discharge end of the riffles there is usually another curve, which serves the purpose of preventing too large a portion of the dressing water from falling over the end of the table with the concentrate. This is for assuring a steady and even flow of water completely across all of the parallel channels of the table below the wash-water box, so as to produce a steady and even flow of water down the channels in the cleaning zone. The terminal curves are indispensable when the riffles have a thickness of from $\frac{3}{16}$ to $\frac{3}{8}$ in. at the discharge extremities of the table.

The stratification and removal of concentrate is so rapid that most of it goes off in the upper one-third of the table deck in 12 to 15 riffles, depending upon the ratio of concentration. The remaining mineral is successively less in volume and is cleaned and brought forward in the larger area farther down the table, and is lighter and successively poorer in grade. Thus there is a large space, between the main concentrate discharge and the silica rejections, which is occupied by a small quantity of relatively poorer concentrate, and there is no large accumulation of concentrate near the middling corner as in the old system. The feed on the middling corner of the table has become so thoroughly impoverished that it is sometimes found necessary to take the contents of several of the lower riffles into the middling.

The effect of the uniform feed and water distribution over this completely riffled table is to produce uniform velocities in the water moving on the two zones each side of the cleaning curves, with an excellent classification of the feed upon the main body of the table. As previously

mentioned, the sands arrange themselves in successively decreasing sizes from the rear toward the middling corner, and the perfection with which this action occurs is found to be an excellent visual guide in judging the work that the table is performing. The better the classifying action noted the better the recovery will be.

In ores affording low ratios of concentration, the sulphide minerals present are usually massive and coarse concentration is usually practiced, and in their anxiety to avoid sliming of the sulphides engineers try to

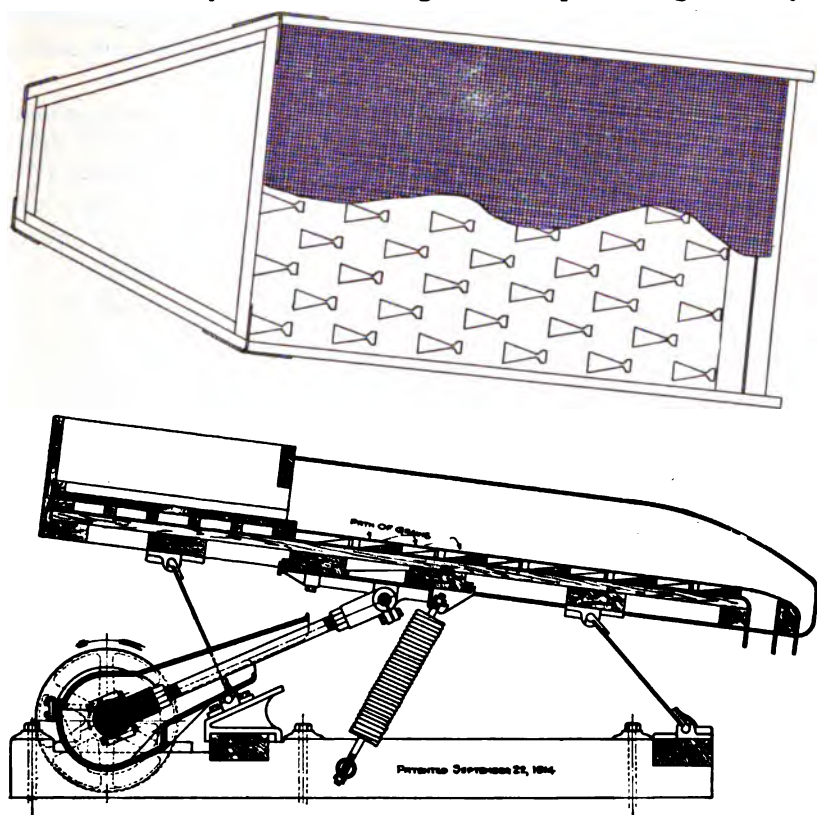


FIG. 5.—VIBRATING SCREEN.

remove them in several stages, each stage adding materially to the complication, cost of construction, maintenance, and cost of operation. The instance quoted in this paper shows that it is not always necessary or economical to have several stages in the roughing-out process, and the author believes that it is seldom advisable.

Roll crushing down to 5 mm. size does not require choke feeds and does not slime the minerals excessively. It is the reduction in regrinders that produces slime rapidly. There is no evidence to show that reducing a $\frac{1}{2}$ -in. cube of sulphide mineral in crushers or rolls into 10 or more frag-

ments passing a 5-mm. screen, in preparation for a single-stage treatment, will result in the production of more colloidal mineral than is produced in the trommels and jigs of a more complex system using more stages, and in the light of the work accomplished and recorded the proposal to simplify the operation by crushing directly to roughing-table size does not seem illogical.

Many copper concentrators have corrosive water to contend with, wherein all iron parts of the equipment coming in contact with the water are subject to corrosion. This makes maintenance of trommels, especially the finer sizes, very expensive, and is an argument against the complex methods commonly in use.

Water is expensive at Morenci and has to be used with great economy. It is kept in circulation as long as possible. The use of lime is resorted to, but the solutions are still corrosive. Wet screens are necessary, and in order to avoid the high first cost and high maintenance expense for these, a vibrating machine for working wet has been devised and very successfully used. As shown in Fig. 5, it is a small machine, carrying in this case a 5-mesh by 0.054 bronze wire screen. It is made of wood, copper, and rubber, the latter being used wherever water comes in contact with it, and, since there is no scouring action from the ore passing over it, the light wire screen mentioned above lasts approximately 100 days, during which time about 40,000 tons of ore pass through the wires. This is about 10 times the life of ordinary wet screens of this mesh. The machines do not clog, because a sluice deck is maintained close to the under side of the screen cloth, and the coarser part of the undersize strikes against the protruding oversize particles and drives them back. On its way down the slope the water also goes back and forth through the screen in a way which is effective in making the separation rapid and in preventing blinding of the meshes. This machine has proved to be a further factor auxiliary to the simplification of flow sheets through the use of the Butchart riffle system.

The results of the improvements are regarded as important in promising to lessen greatly the outlay required to treat a given tonnage in large plants, and by making it possible to assemble a cheap and effective concentrating plant for small tonnages.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Reverberatory Smelting Practice of Nevada Consolidated Copper Co.

BY R. E. H. POMEROY, MCGILL, NEV.

(New York Meeting, February, 1915)

THE statistical data given in this paper are taken from the actual performance of the No. 2 reverberatory furnace of the Nevada Consolidated Copper Co., McGill, Nev., for a period of four months, from April to July, 1914.

The principal furnace dimensions are: Length, 132 ft. $\frac{1}{2}$ in. at the skim-line level; average width, 18 ft., 10 in. at the skim-line level. The roof is 7 ft. above the skim-line level at the firing end. The average area in the uptake flue is 36 sq. ft., and the area above the skim line under the vulcatory arch is 24.3 sq. ft. Fig. 1 shows a diagrammatic plan and sectional elevation of this furnace. The furnace is equipped with two 400-b.h.p. Stirling waste-heat boilers set in parallel.

California crude petroleum is used for fuel and has a specific gravity of 16.5° B., flashing in open test at 199° F. The oil is heated to about 200° F. in a steam heater before going to the burners, requiring about 9 b.h.p. per furnace day to heat it from the line temperature to the burning temperature. Measurement is made by oil meter, checked daily by reservoir readings.

The oil is fed by gravity (34 lb. static pressure) to seven low-pressure blast burners of the Steptoe type (Fig. 2). The blast for these burners is supplied at about 40-oz. pressure by a motor-driven Connersville blower of 42 cu. ft. capacity per revolution.

The air supplied by the blower through the burners for atomizing the oil amounts to about 10 per cent. of the theoretical air necessary for complete combustion of the oil. The remainder of the necessary air enters the furnace through the burner openings in the firing wall. No checker holes are provided, and the charge holes in the roof are equipped with slide gates and "dog houses" to minimize air leakages.

The combustion of the fuel, as shown by gas analysis at the front of the furnace, is practically complete, the carbon monoxide in the gases being less than 0.5 per cent.

The draft at the firing end of the furnace is 0.18 in. of water. At the throat or in the uptake above the "verb" the draft is about 1 in.

of water, while the stack draft is 1.4 in. The top of the stack is about 320 ft. above the skim line of the furnace.

The chart shown in Fig. 3 is a gradient of flame temperatures from the firing end of the furnace to the waste-heat boilers. The temperatures at the side doors were taken with a Scimatco optical pyrometer. That at the verb was taken with a F ry radiation pyrometer, and the temperature in front of the boilers was obtained with a Brown base-metal thermo-couple.

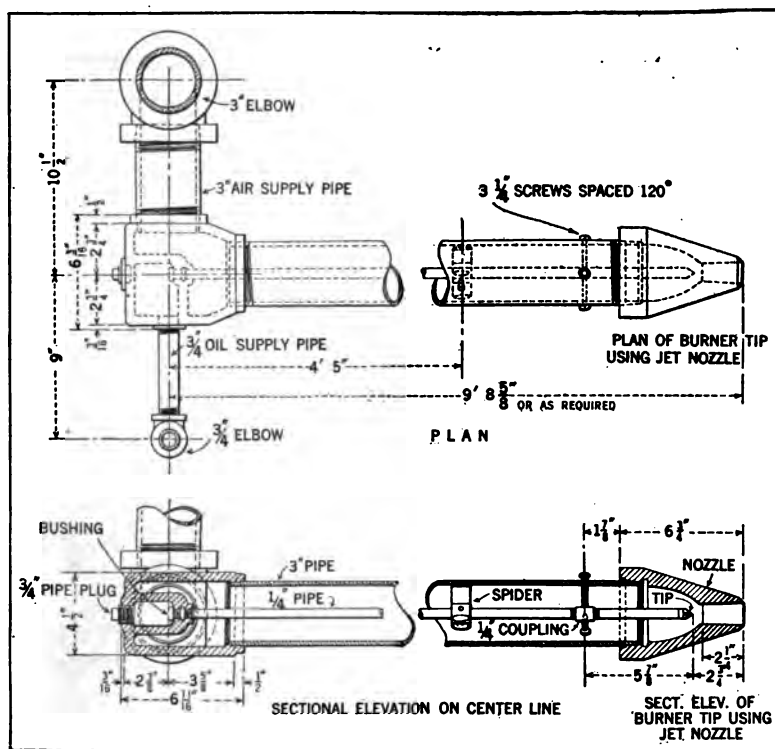


FIG. 2.—Low-Pressure Oil Burners, Steptoe Type.

From this chart the maximum temperature is found near the second side door, about 21 ft. from the burners. The only exception is during the first 10-min. interval after charging, when the combustion is delayed by the cold charge and the inert gases evolved. Here the maximum temperature is advanced about 10 ft. farther down the furnace. The effect of this delayed combustion immediately after charging is also noticeable at the front end of the furnace and in the flue to the boilers, where the temperature is highest during this first 10-min. interval.

The gradient is very uniform and the average temperature range in the furnace is not more than 300° F. between charge and charge.

The sharp drop in temperature between the last furnace door and the boilers is due to the fact that the flame is burned out at the throat of the furnace and only the gaseous products of complete combustion pass on to the boilers. Radiation from the flue and air leakage around the skimming door also contribute to this temperature drop. To minimize this loss of heat the flue has been covered by a coating, 2 in. thick, of asbestos, and the surface painted with heavy asphaltum.

The heat recovery in the waste-heat boilers is based upon the water evaporated, which is measured by water weighers, and is shown as a percentage of an empirical factor of 14 lb. of water per pound of oil. The average over this period was 34.35 per cent., or an evaporation of

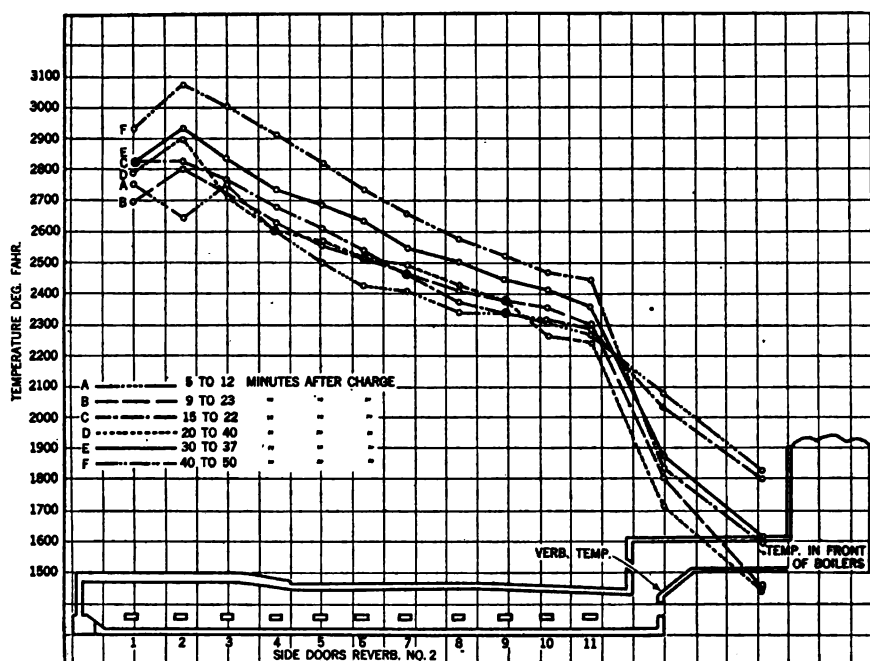


FIG. 3.—CURVES SHOWING SIDE-DOOR TEMPERATURES OF REVERBERATORY No. 2 TAKEN AT DIFFERENT INTERVALS AFTER CHARGING.

4.81 lb. of water per pound of oil burned. The fluctuations in the steam requirements are covered by steam developed in stand-by coal-fired boilers at the power house and no effort is made to regulate the steam output from the waste-heat boilers at the expense of the oil-to-charge ratio. This permits the furnace crews to make the most economical use of the oil for smelting purposes.

Considerable fluctuation in the composition of the concentrates is unavoidable when the ore is mined by steam shovels. The ore must be treated as mined and continuous shovel cuts must be made through

the ore irrespective of its character. The conditions are, therefore, different from those encountered when treating ore from a well-developed orebody, mined by underground methods, where broken ore can be drawn from several stopes to make a desirable smelting mixture after concentration.

The percentage of copper in the concentrates varies from 4 to 10 per cent., and of sulphur from 24 to 34 per cent. The relative quantity of sulphides in the original ore is also a variable factor, hence the ratio of concentration fluctuates over a range of from 4 to 8.5 tons of ore per ton of concentrates.

As the concentrates from the mill are delivered direct to the roasters, no opportunity is offered for effective mixing, except a ground storage pile handled by locomotive crane; the concentrates must be taken to the furnaces as produced, regardless of composition, the fluxing being accomplished by adjusting the lime rock feed.

There being no blast furnace at this plant, it has been our practice for the last five years to make use of all converter secondaries for fettling the reverberatories. (See Table I for analysis.) This material is used more particularly along the firing wall or at some point where heavy material is necessary to sink into the bath and form a secure foundation at the furnace sides for siliceous fettling above and on the slag line. The principal siliceous fettling used is mill slimes or siliceous concentrates. (See Table I for analysis.) The furnaces are fettled daily by hand, using shovel and ladle, and about 30 tons of this material is used per furnace day. No brick side-door coverings are used, as each door is provided with an outside sill or shelf and the opening is piled full of fettling material.

Hot converter slag is poured direct from 10-ton slag cars on the reverberatory charge floor. The molten slag is poured on a steel grizzly and thence through a 16-in. diameter sectional cast-iron pipe into the furnace between the front and back charge hoppers. The skulls or shells from the slag pots are dumped out on the grizzly and broken down into the furnace through the same pipe with the converter slag. Thus the pots return to the converter plant empty and ready for another load.

The chemical composition of the charge components and furnace products is given in Table I.

Table II is a screen analysis of the concentrates.

The charge as fed to the reverberatory is composed of six ingredients, in proportions shown in Table IV.

The performance of the furnace is shown in Table III.

For purpose of comparison with other plants, both total and solid-charge tonnages are shown. The total charge, as the name implies, consists of all the charge, including hot converter slag. The solid charge is the total charge less the hot converter slag. The column in Table III

headed "Net oil ratio" is the "gross oil ratio" corrected for waste-heat credit, calculated as previously explained.

The oxygen ratio of the reverberatory slag, as shown from the analysis in Table I, is about 1.6 acid to 1 of base, with the alumina treated as a base. As a matter of fact, however, the slag is more acid than the analysis indicates, due to the presence of magnetic oxide of iron from the concentrates and calcines, which of course is not available as a flux for silica in a reverberatory furnace.

TABLE I.—*Analyses*

Material	Per cent. Cu	Per cent. SiO ₂	Per cent. Fe	Per cent. CaO	Per cent. Al ₂ O ₃	Per cent. S
Concentrates roasted.....	6.48	28.4	27.0		5.2	28.4
Cold secondaries.....	9.37	19.9	43.4		3.4	8.4
Hot converter slag.....	2.01	27.2	46.2		4.5	1.2
Fettling slimes.....	3.32	65.8	7.3		8.3	7.6
Lime rock.....		2.1	0.8	51.1	0.8	
Roaster flue dust.....	5.85	40.4	15.2	0.7	6.6	9.2
Reverberatory matte.....	27.46	0.4	40.8		0.5	27.2
Reverberatory slag.....	0.298	42.2	28.4	9.5	6.8	0.4

TABLE II.—*Screen Analysis of Concentrates Roasted*

Mesh	+30	+40	+60	+80	+100	+150	+200	−200	Average
Weight, per cent.....	9.7	7.4	13.4	10.4	11.9	13.5	9.6	24.1	
Copper, per cent.....	3.23	4.33	6.16	7.44	7.28	7.60	7.27	6.88	6.48

TABLE III.—*Performance of Furnace*

Month, 1914	Furnace Days	Total Charge per Furnace Day—Tons		Daily Tons per 100 Sq. Ft. Hearth Area Total Charge.		Solid Charge per Furnace Day—Tons		Daily Tons per 100 Sq. Ft. Hearth Area Solid Charge		Oil Fired per Furnace Day—Barrels	Gross Oil Ratio Bbl. per Ton		14-lb. Basis Waste-Heat Recovery, Per Cent.	Net Oil Ratio Bbl. per Ton	
		Total Charge	Solid Charge	Total Charge	Solid Charge	Total Charge	Solid Charge	Total Charge	Solid Charge		Total Charge	Solid Charge		Total Charge	Solid Charge
April.....	30.0	694	29.53	609	25.91	371	0.53	0.61	32.48	0.358	0.412				
May.....	30.8	665	28.29	581	24.72	384	0.58	0.66	34.99	0.377	0.429				
June.....	30.0	659	28.03	582	24.76	372	0.56	0.64	36.43	0.356	0.407				
July.....	31.0	721	30.68	628	26.72	415	0.58	0.66	33.51	0.386	0.439				
Average..	30.4	685	29.15	600	25.53	385	0.562	0.642	34.35	0.370	0.421				

NOTE.—The above period does not cover a complete campaign.

TABLE IV.—*Percentage Analysis of Charge Smelted*

Month, 1914	Calclines	Cold Sec- ondaries	Hot Con- verter Slag	Fettling	Lime Rock	Roaster Flue Dust
April.....	59.3	8.0	12.2	4.6	12.9	3.0
May.....	58.7	8.0	12.6	4.8	12.3	3.6
June.....	63.4	6.6	11.7	5.4	12.1	0.8
July.....	58.4	7.7	12.9	5.3	13.0	2.7
Average.....	59.9	7.6	12.4	4.9	12.7	2.5

The matte fall, to borrow the term from blast-furnace practice, is about 22 per cent. of the total matte and slag produced.

The slag is skimmed off three times during each 8-hr. shift to keep the matte bath as close to the flame as possible and thus facilitate the absorption of heat by the matte from the flame, between charges.

The normal length of a furnace campaign is from 170 to 200 furnace days, and the tonnage smelted during this interval is upward of 110,000 tons. The extent of the repairs to the furnace between campaigns is about as follows:

Renewal of from 20 to 30 ft. of roof measured along the furnace. This roof is 20 in. thick and is located over the hottest portion of the hearth, around the charge holes and near the firing end of the furnace. The side walls are patched where necessary and a little brickwork is usually required at the front end of the furnace. When occasion for haste arises, campaign repairs can be completed in from four to six days between the time of shut down and the time of smelting again.

The original bottom, as shown in Fig. 1, has been worn away until the average depth of the bath below the skim line is approximately 3 ft. As the tap-hole elevation has not been changed, this gives a deep bath of matte below that level, which acts as an accumulator and regulator of heat, much as the flywheel of a steam engine acts to store up and regulate the energy developed in the steam cylinders.

The matte bath is always kept as near the skimming-plate level as possible, the smelting being done on a bath of molten matte about 3 ft. deep. This deep matte bath is of great advantage, as follows:

It serves to equalize and distribute the heat furnished by the flame, thus preventing the overheating of any portion of the hearth, with consequent boiling and damage to the silica bottom.

It becomes superheated, and when a fresh charge is dropped into the furnace it has a tendency to smelt the charge on the bottom while the flame acts on the top, thus increasing the smelting area of the furnace by attacking the charge layer on both top and bottom.

With a deep bath under the charge holes no trouble is encountered

from the charge sticking to the bottom and piling up at one point on the hearth. The charge of from 20 to 25 tons of material is dropped through four holes and spreads out uniformly over the entire hearth, utilizing all of the area under the flame, oftentimes extending down as far as the verb of the furnace.

As the furnace is fettled with great care, no break-outs or runaways of matte, due to working with this deep bath, have yet occurred.

To improve the smelting conditions the lime rock is fed to the MacDougall roasters by an outside feeder, which drops the unscreened crusher product (maximum diameter $2\frac{1}{2}$ in.) on the hearth next above the bottom. It is thus thoroughly mixed with the calcines without cooling the furnace on the upper hearths where the heat is necessary for roasting, and yet is sufficiently dried and heated to prevent excessive dusting in handling.

On entering the furnace the lime rock, mixed with the other charge, comes in contact with the superheated matte bath and is rapidly decomposed with the evolution of CO_2 . The charge spreads quickly over the entire hearth with violent ebullition, which thoroughly stirs and mixes it, effecting the necessary contact of acid particle with basic particle and rapidly absorbing heat from the matte bath below and from the flame above. The coarse lime rock prolongs this boiling action and is more beneficial than the fine-crushed flux.

This uniform and rapid action during the formation of the silicates gives them, when formed, ample time to liquefy and superheat, liberating the particles of newly formed matte and leaving the bath in a hot fluid condition for the next charge. The formation of charge floaters or blankets, due to slow smelting of poorly mixed charge resulting in the smelting out of the more fusible portion of the charge, is thus entirely eliminated, and only an occasional floater of siliceous material is formed from the fettling material which sometimes slips down from the furnace sides and floats away on the bath.

To recapitulate, then, I would say that the vital points in reverberatory smelting, as practiced at Steptoe, are:

Factors affecting efficient combustion of the fuel:

1. The use of a burner to give a proper atomization of the oil so as to obtain a long uniform flame without overheating any portion of the furnace.
2. Regulation of the draft so as to furnish the proper mixture of gases for complete combustion in the furnace.

Factors affecting the operation of the furnace proper:

1. Careful preparation of the charge by adequate mixing of all ingredients before charging.
2. Addition of enough lime rock, preferably coarse, to produce an active boiling in the furnace.

3. Maintaining a deep bath of molten matte to equalize and distribute the heat over the whole of the hearth.
4. Frequent skimming so as to carry only a thin layer of slag over the matte bath.
5. Operating the furnace for the best smelting conditions, ignoring the waste-heat boilers as factors in the power supply.

Factors affecting the life of the furnace:

1. The furnace roof set high over the hottest portion of the hearth.
2. Frequent fettling to protect the side walls.
3. Frequent charging and active charge mixtures to avoid floater and blanket formation requiring excessive firing.

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Sound Steel Ingots and Rails*

BY GEORGE K. BURGESS,† WASHINGTON, D. C., AND SIR ROBERT A. HADFIELD, LONDON,
ENGLAND

(New York Meeting, February, 1915)

1. *Introduction.*—THE methods of production of sound steel ingots have been described in several papers read recently before this Institute. It was thought by Director Stratton, of the U. S. Bureau of Standards, and ourselves to be of sufficient interest and importance, as a further contribution to this subject, to compare the properties of rails rolled in one mill according to American practice, from several types of ingot of substantially the same chemical composition but cast by several processes; and to compare in considerable detail the properties of an ingot manufactured by the Hadfield special-feeding process and one of the usual type for rolling into rails.

We are greatly indebted to F. W. Wood, President of the Maryland Steel Co., who made this investigation possible by kindly placing his rail mill at our disposal; and he and his associates at Sparrows Point did everything possible to make the experiment a success, including the execution of several of the tests and analyses.

2. *Classification and Manufacture of Ingots.*—Of the ingots used in this investigation, eight were made in Sheffield and furnished by Sir Robert Hadfield, and one was made and furnished by an American steel company,¹ as a comparison ingot, and was supposed to represent the usual type of ingot from which rails are made.

The characteristics of the ingots are given in Table I. Ingots 1, 2, 3, and 4 were cast large end up and fed by the Hadfield method in the usual manner with charcoal and blast continued until the molten steel has set on the top of the head, say varying from 20 to 40 min. To ingots 1, 2, and 4 was added 0.1 per cent. aluminum, and to the nickel-chrome ingot, No. 3, 0.125 per cent. For sake of comparison, there were also included two ingots, Nos. 8 and 9, cast with the small end up and fed by

* A complete account of this investigation, giving all data in detail, will be published later as a Technologic Paper of the U. S. Bureau of Standards.

† U. S. Bureau of Standards.

¹ This ingot is not to be assigned to any particular manufacturer, as it is one of several from different sources, supplied without special selection as typical of the usual output of ingots.

the Hadfield process; three ingots not fed, two of piping steel, Nos. 6 and 10, and one of rising or "unsound" steel, No. 7. The Hadfield ingot No. 1 and the American comparison ingot, No. 10, were cut up for examination as to segregation, blowholes, and piping, while the others were rolled into rails. It was expected that ingots Nos. 2, 3, and 4 would give rails of a uniform quality throughout nearly the whole length of the ingot, and that No. 1, which was selected by chance from the specially

TABLE I.—*Classification of Ingots*

No.	Size, Inches	Weight, Pounds	Method of Casting	Type of Steel	Chemical Analysis				
					C	Mn	S	P	Si
1	18	5,964	Hadfield system, large end up.	Ordinary rail steel.	0.59	0.97	0.041	0.031	0.21
2	18	6,006	Hadfield system, large end up.	Ordinary rail steel.	0.55	0.96	0.043	0.029	0.12
3	18	5,900	Hadfield system, large end up.	Nickel-chromium steel.	0.25	0.38	Ni 3.70	Cr 1.47	0.19
4	18	5,850	Hadfield system, large end up.	Ordinary rail steel.	0.56	0.92	S 0.040	P 0.035	0.19
6	18	5,200	Ordinary manner, small end up.	Ordinary rail steel which pipes, not fed.	0.56	0.94	0.060	0.051	0.20
7	18	5,100	Ordinary manner, small end up.	Ordinary rail steel of rising or boiling nature, not fed.	0.57	0.95	0.080	0.045	0.20
8	18	5,700	Hadfield system, small end up.	Ordinary rail steel.	0.58	0.96	0.070	0.051	0.23
9	18	5,700	Hadfield system, small end up.	Ordinary rail steel.	0.59	0.96	0.054	0.040	0.20
10	19		The American ingot, cast in ordinary manner, small end up.	Ordinary rail steel.	0.46	0.94	0.05	0.09	

fed ingots cast large end up, would show a uniform, sound structure throughout. Previous experience has shown that ingots of the type of Nos. 8 and 9, cast by the Hadfield process with small end up, would be expected to be less satisfactory than the Hadfield type represented by Nos. 2, 3, and 4, cast with large end up. The former sometimes show a tendency to develop a pipe at the lower end. Ingot No. 6 would be expected to show piping and segregation, and No. 7 segregation and unsoundness; for both of these ingots a much greater waste would be expected than for the specially fed ingots.

3. *Examination of Ingots.*—At the Pittsburgh Laboratory of the Bureau of Standards two of the ingots were sawed in halves—Hadfield No. 1, of the specially fed type cast large end up, and the comparison ingot, No. 10, from an American mill. Figs. 1 and 2 are photographs of the two ingots showing the location of drillings and a comparison of the interiors of the two ingots. The Hadfield ingot was also smoothed off in order to take a sulphur print, shown in Fig. 3. In Figs. 4, 5, and 6,

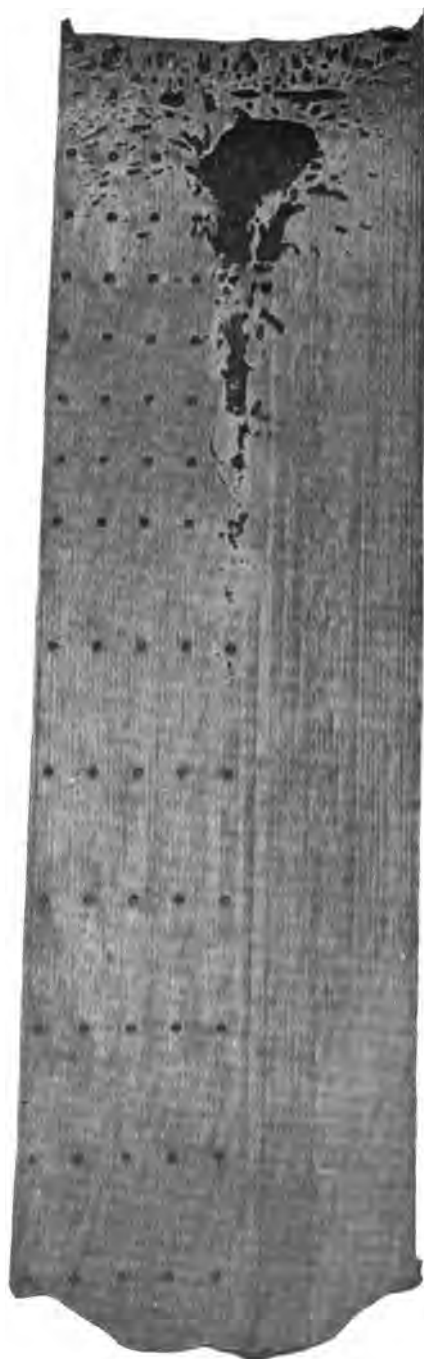


FIG. 1.—SPLIT HADFIELD INGOT.

FIG. 2.—SPLIT COMPARISON INGOT.

FIGS. 1 AND 2.—SHOWING LOCATION OF DRILLINGS AND CONDITION OF THE INTERIOR OF THE INGOT.

TABLE II.—*Rolling and Inspection of Ingots and Rails*

	3	2	4	6	7	8	9
Number of ingot	8 hr. 9 min.	8 hr. 40 min.	8 hr. 55 min.	9 hr. 11 min.	9 hr. 27 min.	9 hr. 43 min.	10 hr. 45 min.
Taken from pit at	1 min. 40 sec.	1 min. 40 sec.	1 min. 42 sec.	1 min. 30 sec.	1 min. 40 sec.	1 min. 30 sec.	1 min. 40 sec.
Time in blooming mill	1,100°	1115°	1,110°	1,110°	1,135°	1,150°	1,125°
Average temperature of ingot in blooming mill, deg. C.							
Time blooming mill to hot saw.	3 min. 50 sec.	4 min. 10 sec.	3 min. 23 sec.	4 min. 5 sec.	3 min. 20 sec.	3 min. 15 sec.	3 min. 22 sec.
Average temperature first rail bar at hot saw.	950°	960°	970°	970°	970°	980°	980°
Average temperature second rail bar at hot saw.	945°	900° ^a	950°	945°	955°	970°	965°
Top bloom butt, per cent. of ingot.	4.2 ^b	5.6 ^b	7.8	12.9	5.0	5.1	6.3
Total bloom butts, per cent.	7.2	7.4	9.7	14.7	8.0	6.8	8.3
Oxidation loss, per cent. ^d	1.3	2.7	1.9	2.9	3.5	3.5	2.5
Percentage of ingot available for rails.	91.5	89.9	88.4	82.4 ^e	88.5	89.7	90.2
Rails: Percentage of firsts.	77.2	77.1	52.5	9.5	54.3	51.9	28.5
Percentage of seconds/ ^f	22.8	22.9	47.5	90.5	27.4	48.1	23.8
Percentage of scrap/ ^f	0	0	0	0	18.3	0	47.7 ^e
Type of ingot	Special	feed, large end up	end up	Ordinary piping	Ordinary rising	Special feed, small end up	

^a Was held back before coming to saw.

^b Has been corrected as explained in §4.

^c Contains pipe, however, extending into C rail.

^d It is to be kept in mind that all are reheated ingots.

^e Two rails, C and D, from middle of ingot were scrapped for mechanical defects.

^f This classification includes, of course, defects due to mill practice.

respectively, are shown the segregation of carbon, phosphorus, and sulphur for each of the ingots. A similar survey was also made for manganese, with no considerable segregation of this element in either ingot. It is evident that there is freedom from appreciable segregation over 95 per cent. or more of the Hadfield ingot, which is also entirely free from piping or blowholes. The comparison ingot, if rolled into rails, should have about a 50 per cent. discard.

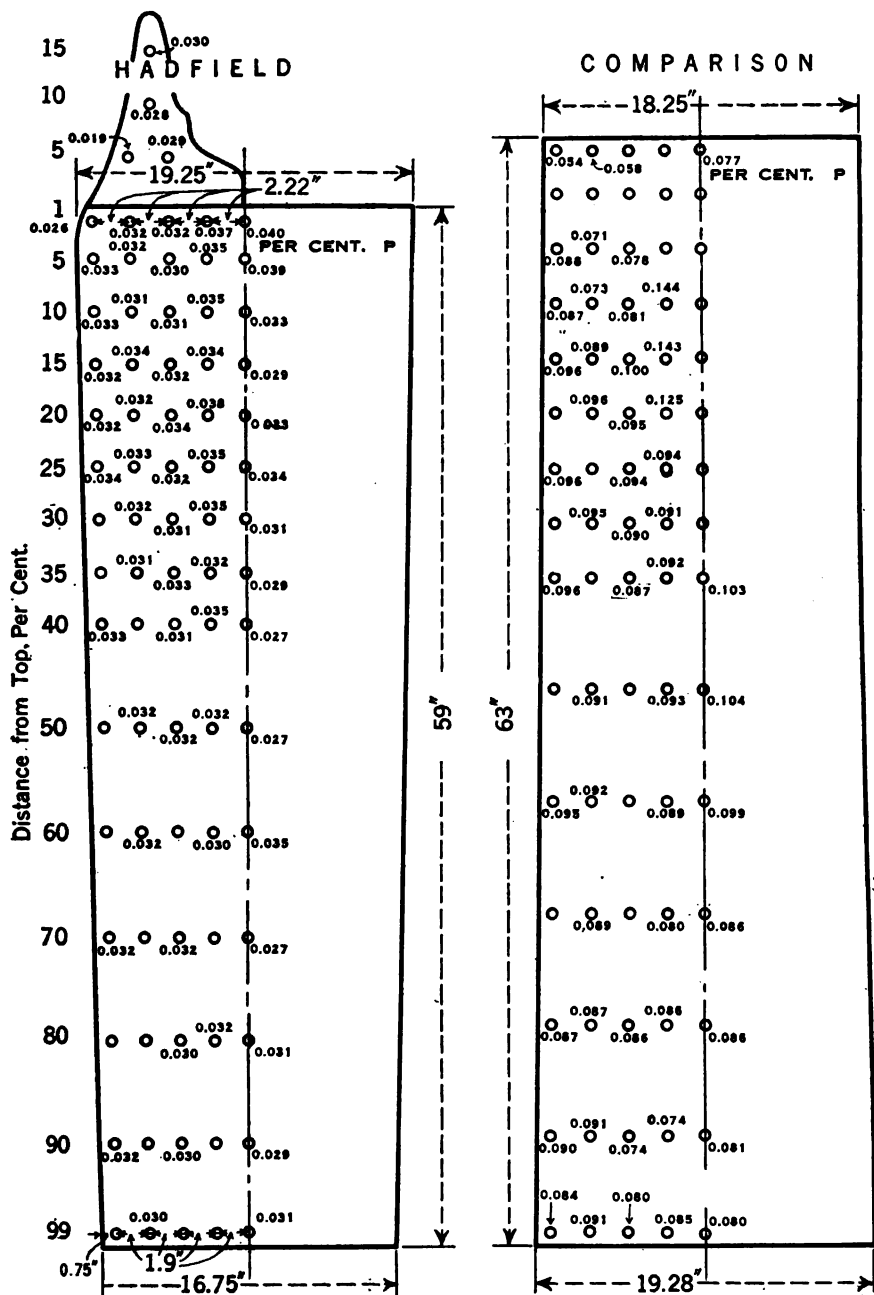
4. *Manufacture of Rails.*—The ingots were charged into the soaking pits, which had been allowed to cool considerably, on Sunday evening, and rolled Monday, Apr. 13, 1914, at 8 A. M., into 100-lb. P. S. section rails, the seven ingots being interpolated among other ingots. In the Maryland mill the rolling from ingot to finished rail is a continuous process without reheating of the blooms. In Table II are shown some of the characteristics of the rolling. Apparently through a misunderstanding of instructions and unfamiliarity with this type of ingot, the top bloom crops of the first two rolled, specially fed Hadfield ingots Nos. 2 and 3, were excessive, at least double what was necessary to carry the blooms through the rail mill. The other Hadfield and comparison ingots were sheared as instructed. The A rail was made the short rail, usually 6 to 10 ft. long, to be examined as to soundness. The drop-test specimen was cut next the bottom of this short A rail, a 13-in. specimen for tensile tests from the head of the C rail, and 3-in. pieces for chemical, metallographic, and hardness tests from the heads of the A, B, and C rails.

5. *Tests of Rails.*—In Table III are shown the Brinell hardness tests taken at the top of the A and C rails for positions shown in Fig. 7. It is of interest to note that the tests, with the exception of those on rails with a pipe or seam (from ingots Nos. 6, 8, 9), show a high value of hardness numeral for position 5 in the A rail. Otherwise the hardness is quite uniform over the sections, and from one rail to another, except, of course, for the nickel-chromium, No. 3.

The results obtained by the Maryland Steel Co. for the drop tests are shown in Table IV; of inspection

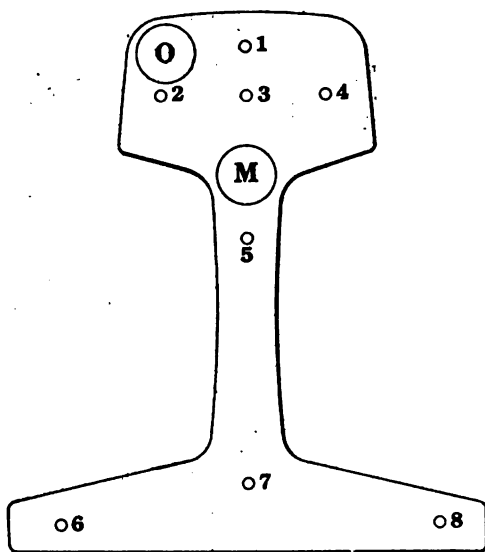


FIG. 3.—PHOTOGRAPH OF SULPHUR PRINT OF HALF-SECTION OF HADFIELD INGOT.



tion in Table II; physical tests in Table V; and chemical analysis for carbon in Table VI in which the *O* and *M* positions refer to Fig. 7. The *O* and *M* analyses for all the elements and for the *A*, *B*, and *C* rails will be given in the complete paper.

Figs. 8, 9, 10, and 11 are characteristic sulphur prints of rail sections for each type of ingot. For all the Hadfield ingots cast large end up the rail macrostructure is very uniform even over the web for the front end of an *A* rail, as here shown. Rails from both the Hadfield ingots cast small end up show a very different macrostructure from the first group. The center streak of Fig. 11 has the appearance of nearly pure ferrite under the microscope; this corresponds also to the hardness examination of Table III. It is interesting to note that the pipe of ingot No. 6 cast in the ordinary way is not accompanied by any considerable segregation. This pipe extends well into the *C* rail even after a top bloom



O and *M*, positions of samples for chemical analysis.
1-8, positions of hardness tests.

FIG. 7.—RAIL SECTION.

discard of 13 per cent., confirming the characteristics of the split comparison piping ingot No. 10 of §3. In the complete paper the microstructure will also be reported in detail.

From the several tables of tests and from the metallographic examination it is seen that the percentage of ingot available for sound rails of uniform homogeneous structure is 91, 90, and 88 respectively for the specially fed ingots cast large end up, Nos. 2, 3, and 4, as compared with much less percentages of sound ingot metal usually available.

6. *General Conclusions.*—In all, nine ingots were used in this investigation; eight furnished by Sir Robert Hadfield, all but one of which had approximately the same chemical composition.



FIG. 8.—SULPHUR PRINT OF SECTION FROM FRONT END OF A RAIL FROM INGOT No. 2, SPECIALLY FED, CAST LARGE END UP.



FIG. 9.—SULPHUR PRINT OF SECTION FROM FRONT END OF A RAIL FROM INGOT No. 6, CAST IN ORDINARY MANNER FROM PIPING STEEL. THE PIPE SHOWN EXTENDS INTO THE C RAIL.



FIG. 10.—SULPHUR PRINT OF SECTION FROM FRONT END OF A RAIL FROM INGOT NO. 7, CAST IN ORDINARY MANNER FROM RISING STEEL.



FIG. 11.—SULPHUR PRINT OF *B* RAIL FROM INGOT NO. 8, SPECIALLY FED, CAST SMALL END UP.

TABLE III.—*Brinell Hardness Numerals*

Ingot	Rail	Hardness Numeral in Kg. Per Sq. Mm. for Positions								Mean	Average Deviation
		1	2	3	4	5	6	7	8		
2	A	250	253	250	251	272	258	258	251	255	± 5.4
2	C	258	254	253	259	260	259	256	254	257	2.4
3	A	366	347	374	351	398	362	359	359	365	11.2
3	C	351	351	364	351	360	351	351	351	354	4.2
4	A	244	241	253	247	295	246	252	250	253	10.0
4	C	251	251	253	254	253	251	251	251	252	1.1
6	A	248	244	254	251	251	251	252	241	249	3.5
6	C	248	252	256	250	245	246	255	252	251	3.2
7	A	248	250	262	256	276	251	245	239	253	8.5
7	C	248	260	254	262	251	251	255	248	254	4.1
8	A	250	248	260	248	218*	258	260	254	250	8.5
8	C	254	258	264	251	279	261	263	263	262	5.6
9	A	248	251	251	245	236*	255	264	260	251	6.2
9	C	260	258	260	262	264*	264	264	263	262	1.9

* Seam through indentation.

TABLE IV.—*Drop-Test Results*

100-lb. P. S. section; 3-ft. supports; weight of tup, 2,000 lb.; fall, 15 ft.

Ingot No.	Blow	Per. Set	Elongation on base						
			1st in.	2nd in.	3rd in.	4th in.	5th in.	6th in.	Total
2	First.....	1.1	0.03	0.04	0.05	0.05	0.04	0.03	6.24
3	First.....	0.8	0.03	0.04	0.05	0.05	0.04	0.03	6.24
4	First.....	1.1	0.02	0.04	0.05	0.05	0.04	0.02	6.22
6	{ First, pipe at fracture.	1.0	0.02	0.03	0.04	0.04	0.03	0.02	6.18
7	First.....	1.0	0.02	0.03	0.04	0.04	0.04	0.03	6.20
	Second.....	1.9	0.03	0.04	0.06	0.06	0.06	0.06	6.31
8	{ First, pipe at fracture.	1.0	0.02	0.04	0.05	0.05	0.03	0.03	6.22
9	First.....	1.0	0.02	0.02	0.05	0.05	0.04	0.03	6.21

The examination of the split ingots shows the great superiority of casting large end up and feeding by the Hadfield special process, such an ingot being physically sound and uniform throughout and practically free from chemical segregation.

TABLE V.—*Results of Tensile Tests Cut from Center of Heads of C Rails*

Ingot No.	Yield Point* Lb. Per Sq. In.	Ultimate Strength, Lb. Per Sq. In.	Elongation Per Cent. in 8 In.	Reduction of Area	Fracture, Per Cent.
3	98,700	177,300	9.75	27.3	100 silk, cup and cone
2	69,100	119,800	14.00	25.2	10 silk
4	69,000	118,800	12.00	26.0	10 silk
6	69,300	119,300	14.00	27.3	10 silk
7	67,000	120,800	11.75	23.6	10 silk
8	69,000	119,800	12.00	20.4	10 silk
9	68,000	119,000	11.75	20.4	10 silk

Ingot No.	Analysis ^b						
	C	Mn	P	S	Si	Cr	Ni
3	0.268	0.41	0.034	0.043	0.195	1.44	3.65
2	0.562	0.99	0.033	0.047	0.169		
4	0.570	0.90	0.040	0.046	0.197		
6	0.580	0.90	0.052	0.054	0.197		
7	0.618	0.93	0.053	0.066	0.197		
8	0.598	0.94	0.041	0.070	0.216		
9	0.610	0.98	0.044	0.057	0.226		

* Yield point found by use of dividers and running the machine back until the point was found where there was a permanent stretch.

^b Analyses shown are from drillings taken from pulled tensile specimens.

TABLE VI.—*Carbon Analysis of Rails to Show Segregation*

O and M positions of Fig. 7 for tops of A, B, and C rails

Ingot No.	A-O	A-M	B-O	B-M	C-O	C-M	Method of Casting
2	0.540	0.612	0.560	0.556	0.570	0.588	} Large end up; special feeding.
3	0.278	0.290	0.250	0.270	0.240	0.272	
4	0.550	0.640	0.560	0.580	0.612	0.580	
6	0.528	0.584	0.536	0.552	0.560	0.558	Small end up; ordinary piping steel.
7	0.550	0.648	0.570	0.598	0.578	0.568	Ordinary rising steel.
8	0.530	0.640	0.570	0.662	0.578	0.588	} Special feeding.
9	0.598	0.640	0.580	0.600	0.576	0.588	

The examination of the rails shows that for ingots cast by the Hadfield process, those cast large end up will give rails of uniform quality free from flaws of all kinds; those cast small end up may show a soft region in the

web of the A rail but are otherwise sound. Those cast in the ordinary manner from piping or otherwise unsound steel may or may not give sound rails, depending upon the condition of the steel.

In addition to the certainty of producing all sound rails by the special-feeding process from ingots cast large end up, there is a great saving in metal and a consequent increase in percentage of sound ingot available for rails—in these experiments an average of 90 per cent. for ingots Nos. 2, 3, 4, as compared with about 50 per cent. for the piping ingot and an uncertain amount from the rising type.

The usual physical and chemical tests do not appear to give an adequate measure of the quality of rails. For example, the results of the tensile and drop tests and chemical analyses would not separate the sound from all the unsound rails here examined.

The question may be asked, Is it not worth while to use, in the manufacture of rails, only sound steel which is cast and rolled in such a manner as to make practically every rail a sound and safe one? We believe this can be done without excessive cost.

We are greatly indebted for valuable assistance from several members of the staff of the Bureau of Standards, particularly Messrs. Hillebrand, P. H. Bates, Rawdon, Witmor and Devries.

We also take this opportunity of thanking Mr. Milne and Mr. Dawson, of Sheffield, England, for their share in this research and for the care and attention bestowed by them upon the work.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Estimation of Oil Reserves

BY CHESTER W. WASHBURN, NEW YORK, N. Y.

(New York Meeting, February, 1915)

At present it is impossible to estimate closely the amount of oil obtainable from a given area of land. However, after the completion of a few properly distributed prospect wells, one can calculate the approximate yield of the sands penetrated, with a probable error of say 50 per cent. Even rough predictions of this kind are of value in large operations.

Without wells upon it, one can never be absolutely certain that an area will produce any oil whatever, and one can only indulge in wild guesses of the probable productive area, before the geological structure has been carefully contoured and the top of the basal water plotted on a map. This requires wells.

At times it is desirable to form an idea of the possible yield of oil in advance of drilling, as in cases where the price asked for undeveloped land seems too high, but where the probability of finding oil is furnished by neighboring development, supported by the geology. In such cases the most one can do is to make an estimate of possible maximum yield, which may of course far exceed results, and which is of use mainly in the prevention of excessive initial investment.

The method is very significant in a general way, as in regions like the Bighorn Basin, Wyoming, where the productive sands in the Cretaceous shale are so few and so thin and fine that they cannot develop very great production, although the high quality of the oil probably will counterbalance this defect sufficiently to give local profit in favorable areas.

Reservoir Capacity of Sands.—The porosity of a stratum is the measure of its maximum reservoir capacity for liquids and gases. The porosity may be determined experimentally and must be used in connection with the total volume of the sand.

Volume of a Horizontal Sand, per Hectare and per Acre

Area	Thickness of Sand	Volume of Sand	
		in Cubic Meters	in Barrels of 42 Gal. (U. S.)
1 hectare	1 meter	10,000	62,898
1 hectare	1 foot	3,048	19,171
1 acre	1 foot	1,233	7,758

The figures of the last column multiplied by the thickness of the sand, by the porosity, and by the relative saturation, give the capacity of the sand in barrels per unit area. Thus, a sand 12 ft. thick, with a porosity of 15 per cent. and a relative saturation of 75 per cent., contains $12 \times 0.15 \times 0.75 \times 7758 = 10,473$ bbl. per acre. Assuming an extraction factor of 60 per cent., each acre would produce 6,284 bbl.

Under the metric system, in foreign fields, one would apply the third column, with a correction for specific gravity. Thus a sand 20 m. thick, with a porosity of 20 per cent. and a relative saturation of 60 per cent., would contain $20 \times 0.20 \times 0.60 \times 10,000$ or 24,000 cu. m. of oil per hectare. If the oil had a specific gravity of 0.900 this would equal $0.900 \times 24,000$ or 21,600 metric tons. With 60 per cent. extraction each hectare would produce 12,960 tons of oil.

Values of Factors.—The uncertainty of these figures becomes evident when one considers the doubtful factors on which they are based.

The relative saturation depends mainly on the volumes of free gas and of water which are occluded in the oil or entangled in the oil-bearing layers. It can be estimated by analysis of the product of wells in the same region that occupy analogous positions, allowance being made for the volume of gas emitted, which should be reduced to its volume under the pressure in the sand. In the absence of experiments the writer arbitrarily estimated the relative saturation at 75 and 60 per cent. in the preceding examples, the volumetric remainders of 25 and 40 per cent., respectively, being assigned to gases and water. In some fields the relative saturation decreases to zero with time, and the exhausted wells produce only water accompanied by more or less gas of some kind.

The porosity of sands varies from nearly zero to over 20 per cent., changing greatly in different parts of the same sand. Many field determinations are required to determine this value. The writer suspects that if the porosity be determined from specimens gathered on the outcrop, it will be too low, because in a recent experiment the porosity of a deeply buried sand was about one-fourth greater than the porosity of the same sand at the surface. The difference appeared to be due to the slight deposition of mineral matter (largely calcite) in the interstices of

the surface specimens. The grains of the surface specimens appeared to be identical in size, shape, etc., with those from a depth of 1,600 ft.

Moreover, underground joints and fissures may greatly increase the storage capacity of a sand. The volume of fissures in any rock and of solution cavities in limestone or dolomite is incapable of estimation. Hence the oil reserves of Mexico, a large part of which lie in cavities of this kind, cannot be estimated with anything like the approximate value that holds for Arnold's estimate of the oil reserves in the sands of California.

Extraction Factor.—The preceding figures approximate the amount of oil contained in a so-called saturated sand. Not all of this oil can be recovered under the present methods of exploitation, and the preceding result must be multiplied by a fraction representing the part of the oil which can be taken out of the ground through oil wells. This fraction, which may be called the extraction factor, is thought to vary from 60 to about 80 per cent., the higher figure being probable only in the case of gas-rich oils in sands which have been pumped to a vacuum.

The amount of oil left in the so-called exhausted pools of the Eastern States is probably a noticeable fraction of their production, as indicated by the success of the new method of forcing compressed air into wells in order to drive the remaining oil through the sand to neighboring wells, where it is pumped. The method is clearly better than that of extreme sucking with pumps after the decline of the wells, because at that stage most of the oil lies in the finer pores, as for instance in the sharper corners between the sand grains, etc., and it requires a considerable pressure gradient to move oil in such places. The vacuum method of pumping is undesirable because it does not create a high enough pressure gradient, much of the "vacuum" being overcome in the sand by the progressive vaporization of the light gasoline. This vaporization renders the little masses of oil more viscous and harder to move. The vacuum method, while probably best for worn-out gas sands, is not adapted to the requirements of old abandoned oil fields, which can best win a temporary new lease of life by the application of compressed air. In this way one can obtain all of the oily products that are susceptible of extraction, except the residual gas. It is the most economical in the end, and the most consistent with the principles of conservation.¹ It seems not unreasonable to assume that the application of the method would add 5 per cent. to the total production of a field of light oil.

The entire question is one of practical importance, and the problems involved can be solved with much greater accuracy. All we need is more data, which can be obtained through statistical studies by the U. S. Geological Survey and the U. S. Bureau of Mines.

¹ I am told that L. Dunn of Marietta, Ohio, was the first to apply this method for the rejuvenation of old pools.

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Effect of Zn_3Ag_2 upon the Desilverization of Lead

BY F. C. NEWTON, MAURER, N. J.

(New York Meeting, February, 1915)

REFINERS of lead by the Parkes process have always been solicitous of recovering the zinc used in the desilverization, and justly so, as the loss in zinc constitutes one of the heavy costs in this method of refining. Part of this loss is due to the absorption of zinc by lead, and in the present state of knowledge is not recoverable as metallic zinc. The remainder of the zinc, which finds its way into the precious-metal-bearing crust, is recoverable and naturally engages the operators' most exacting attention.

Careful elimination of the impurities in base bullion has enabled the refiner to produce a zinc crust better adapted to retorting, which reflects favorably in the zinc recovery; and to produce a richer crust, thereby lowering the zinking cost, as the increased efficiency of the zinc results in less of it being in circulation.

It is obvious that if the ratio of concentration be increased, a lessening of the amount of crust formed will ensue, throwing less work on the retorts and diminishing the amount of retort metal sent to the cupels, thus making an actual money saving in all these operations.

In the effort to obtain a rich crust, the metallurgist has increased the heat to a considerable degree, upon the assumption that at the higher temperature the zinc-silver alloy could be squeezed free from the adhering lead.

A further possibility for improvement suggested itself in the freezing-point curve of Prof. H. C. Carpenter, upon zinc and silver. Study of this curve disclosed the existence of a definite chemical compound, Zn_3Ag_2 , with a freezing point of $665^{\circ} C.$, a promising compound for practical application.

In the experiment undertaken it was endeavored to force this compound into the crust, hoping thereby to obtain a higher concentration than formerly. It was recognized that in bringing a large kettle of lead to such a heat, more fuel would be used, the life of the kettle would perhaps be shorter, and other disadvantages would be experienced. As a commercial operation, the project would have to stand on its own feet,

so the records were carefully kept to balance the advantages against the disadvantages.

The first experiment, however, showed such refinement unnecessary, as it was conclusively indicated that the melting point of the zinc-silver compound was so modified as to nullify its feasibility as a commercial factor.

In the first experiments the bullion in the kettle was heated to 705°C ., 40° above the melting point of the zinc-silver compound to be formed, and the zinc stirred in as rapidly as possible. Great care had to be taken in adding the zinc to the red-hot bullion, as in more than one instance the zinc took fire and spoiled the experiment. The work about the hot kettle was very trying to the men, and had we been successful in the endeavor to make a rich crust, it is doubtful if we could have held our kettle crew to the work. After the zinc was stirred in, the kettle was cooled, the crust removed in approximately 55°C . stages with the Howard press, and squeezed at a pressure of 90 lb. per square inch. The crusts removed at the various stages were retorted separately, a dip sample being taken from the retort metal and carefully assayed. All conclusions are based upon the assay of the rich bullion remaining in the retort after distillation, and referred to as retort metal. Reasonably accurate sampling of the rich zinc crust is practically impossible.

It will be seen in the following experiments that, in every instance, the crust removed at the highest temperature, that is, just below the freezing point of Zn_2Ag_2 , furnished the lowest retort metal.

Experiments

Series A

Doré contents of bullion, 191 oz. Zinc stirred in at 705°C .

	Removed at $^\circ\text{C}$.	Retort Metal, Ounces
First press.....	650	560
Second press.....	595	1,413
Third press.....	540	1,537
Fourth press.....	485	2,090
Fifth press.....	430	3,290

Series B

Doré contents of bullion, 253.8 oz. Zinc stirred in at 705°C .

	Removed at $^\circ\text{C}$.	Retort Metal, Ounces
First press.....	595	1,117
Second press.....	540	2,224
Third press.....	485	3,059
Fourth press.....	430	3,944
Fifth press.....	400	3,418

Series C

Doré contents of bullion, 315.5 oz. Zinc stirred in at 705° C.

	Removed at ° C.	Retort Metal, Ounces
First press.....	625	1,864
Second press.....	510	4,466

These tests clearly demonstrated the impracticability of removing the crust at a very high temperature. Further experiments were made to determine what beneficial result, if any, would be obtained by the greater heat. In these experiments, two kettles of metal of the same doré contents were desilverized, one heated to 705° C., and the first press removed at 535° C.; the other treated according to the normal procedure, the zinc being stirred in at 535° C. and the pressing begun immediately.

Series D

Doré contents of bullion, 263 oz. Zinc stirred in at 705° C.

	Removed at ° C.	Retort Metal, Ounces
First press.....	535	4,231
Second press.....	485	4,627
Third press.....	430	4,212
Fourth press.....	400	3,900
Fifth press.....	370	4,076
Sixth press.....	345	2,818
Average		4,100

Same bullion stirred at 535° C., pressed at 535° C. = 3,210 oz.

Series E

Doré contents of bullion, 315 oz.

Stirred in at 705° C.; pressed at 535° C.; retort metal, 4,165 oz.
Stirred in at 535° C.; pressed at 535° C.; retort metal, 4,465 oz.

Series F

Doré contents of bullion, 318.4 oz.

Stirred in at 680° C.; pressed at 535° C.; retort metal, 5,011 oz.
Stirred in at 535° C.; pressed at 535° C.; retort metal, 5,505 oz.

In all cases where the greater heat was used for stirring and pressing, the dross and blue powder formed, especially the former, were far in excess of current practice.

The accompanying curves (Fig. 1), to some extent, show the effect of the temperature. The bullion desilverized in the various experiments was of different doré contents. It is a well-known fact that a richer crust will result from high-grade bullion upon desilverizing than from low

grade. I have endeavored to compensate for this, in the curve, by converting the doré contents of the retort metal, in proportion to the assay



FIG. 1.—CURVES SHOWING INFLUENCE OF TEMPERATURE ON DORÉ CONTENTS OF RETORT METAL.

value of the bullion treated. A sudden drop in the grade of the retort metal at the lower temperature is due to the increase of lead contents, the lead solidifying too quickly to be squeezed from the zinc-silver compound.

The reasons for the fact that desilverization at high temperature gives a lower retort metal are not quite clear. Alder-Wright and Thompson refer to the solubility of the zinc-silver compound in lead, also its dissociation by means of lead. Kremen and Hofmeier in a paper upon Ternary Systems of Lead-Zinc and Silver-Zinc Compounds, base their investigations upon the old freezing-point curve of Petrenko. They overlooked, however, the point vital to the practical application, namely, the solubility of the zinc-silver compound in lead, or its desilverization by means of lead. A full explanation of the results obtained in the above experiments would require an extensive laboratory research upon the behavior of Zn_2Ag_2 with lead at varying temperatures. While this would be interesting, we could hardly alter the facts.

The conclusion to be drawn is, that as a commercial project, the slight increase in doré contents of the crust, which might be obtained from zincking at the greater heat, is more than counterbalanced by the attendant disadvantages. The current practice of to-day, dictated more or less by the question of fuel economy and better life of kettles, has been justified as a practical and efficient procedure from the standpoint of concentration. For comfort of operation, effect upon the men, etc., all arguments are in favor of zincking at a lower temperature.



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Mining Methods at Park City, Utah

BY JAMES HUMES,* PARK CITY, UTAH

(New York Meeting, February, 1915)]

THE active mines in the Park City district at the present time are the Silver King Coalition, Daly-Judge, Daly West, and Silver King Consolidated. Several other companies, such as the Daly, American Flag, Ontario, etc., are doing considerable development work.

In the Silver King Coalition mines the ore is a replacement of a dark siliceous limestone bed, varying in thickness from 18 in. to several feet. In places the ore has reached a thickness of 25 ft. This limestone bed lies within 100 ft. of the underlying Weber quartzite. There are other lime beds, similar in appearance to this one, in the Park City formation, which is approximately 700 ft. thick; but not one of them has been found to contain any traces of ore. Directly under and in contact with the ore-bearing bed is another, composed of gray lime well studded with dark concretionary nodules; and this one is an unfailing indication of the proximity of the ore-bearing bed.

These ore deposits make out from or have connection with some one of the ore-bearing fissures, which were undoubtedly the channels for the circulation of mineral-bearing solutions. It is therefore proper to run prospecting drifts in the fissures, although they seldom contain much ore.

As a general rule, drifting in the fissures is not costly. Mining and mucking cost from \$2.50 to \$4 per foot, and very little timber is required. There are instances, as in the Daly fissure in the Daly-Judge mine, where the expense of keeping drifts open in the fissure is so great that it has proved economical to make the drifts in the foot-wall rock and run cross-cuts into the fissure at short intervals.

In the Park City mines it is not easy to determine beforehand the best method for extracting the ore from a newly discovered ore shoot in the bedded deposits. While we may get one dimension from the drift run in the fissure, that will not give us a correct idea as to the other dimensions of the orebody. The fissures here referred to have a N. 35° E.

* Superintendent of the Silver King Coalition Mines Co.

strike and the beds dip from 10° to 30° NW. The accompanying sketch map, Fig. 1, is a tracing showing part of a worked-out orebody, and has a connection with an ore-bearing fissure at the southeast end and 200 ft. higher up on the dip of the beds. It will be noticed that the 900 drift was advanced a long way in barren rock, yet it was all the time in the replacement bed, and where the ore is shown at the top of the incline it was very thin and discouraging, but persistent exploration developed a fine body of ore.

The orebodies are somewhat lenticular in form, with their greatest length along the strike of the lime beds. The perimeter, except on the fissure side, is very irregular in outline. There are numerous branches, extending sometimes as much as 100 ft., and averaging from 5 to 25 ft. in width. We also find the ore making up and down in small fissures for a few feet; and we encounter "rolls" in the foot-wall that make the beds almost vertical for from 5 to 25 ft.

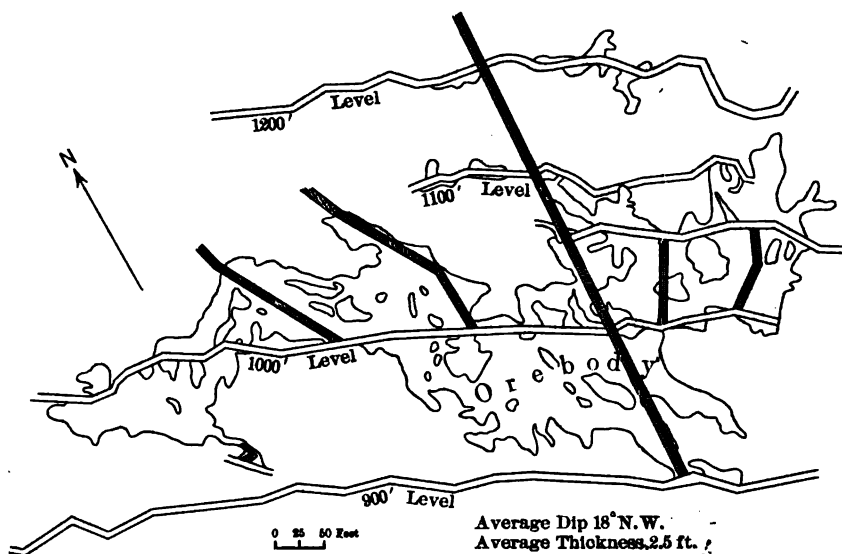


FIG. 1.—SKETCH MAP OF WORKED-OUT OREBODY.

Most frequently the ore dips down from the fissures (that is, the north-dipping fissures); and if our drift in the fissure has cut the ore we work down on it as far as we can by shoveling the rock out. If it persists in going down, we rig up one of our many small hoists, of which we have over a dozen, ranging in size from a Holman stretcher-bar hoist to a double-cylinder 35 h.p. These little hoists enable us to raise our regular mine cars. If the angle is too steep for a direct pull we simply fasten the end of the hoisting rope to a post at the top of the inclined winze, and pass it through a 7-in. pulley which is hooked into the car. This method gives

the engine a "double purchase;" and even with our little double-cylinder 5 by 6 in. engines we are seldom unable to manage our 18 cu. ft. cars. If the deposit is too thin, we take up enough of the bottom to admit the car. Very often our temporary inclines will meet with a face of barren rock. In such cases we do not cut through the waste but turn our rails to the right or left, as the case may be, and follow the ore. (See sketch map.) Should the ore continue down on the dip for a considerable distance, we run small drifts right and left from our main incline, in which we use a small special car, dumping its contents into a hopper that connects with the car on the main incline. Should there be a sufficient thickness of ore at the point where we wish to run one of these sublevels, we simply put down a track switch on the main incline, the points of which we throw with a lever on the level that is connected with the points by means of a long rod, crank, and short rod, and in this way we are able to load the mine cars that are hauled to the ore bins on the surface.

An experienced miner, reading these lines, will probably ask himself the question, "Why don't they do their development work from the lower edge of their orebodies?" As I said before, it is impossible to predetermine the outlines of such orebodies. When we took charge of this property we had many schemes for moving the ore from the faces to the levels. The best and easiest of application was the shaking launder, but as time went on we abandoned the thought of trying any of them. At the present time we do as much "underhand stoping" as overhead stoping, and in doing so we avoid doing a great deal of development work. Most of the time the deposit is so flat and thin that it is impossible to use a stoping machine to advantage, so that most of our mining is done by the "single jack" method. Sometimes the ore gets down to a few inches thick, but we do not abandon it until there is no sign of ore; for many times a small stringer has led us to a large deposit. The cost of mining ore here ranges from \$1 to \$4 per ton.

We believe that the above methods prevail in all the mines here where they are mining the replacement deposits. Following is a scale adopted by the Daly-Judge Mining Co. for running drifts and crosscuts with Leyner machines, the company furnishing all supplies:

Average Footage per Shift	Machine Men	Machine Helpers	Muckers
Less than 3.5	\$3.25	\$3.25	\$3.00
3.5 to 4.0.....	3.50	3.25	3.00
4.0 to 4.5.....	3.75	3.50	3.00
4.5 to 5.0.....	4.00	3.50	3.00
5.0 to 6.0.....	4.25	3.75	3.25
6.0 to 7.0.....	4.50	4.00	3.25



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Mining Methods of the Arizona Copper Co.

BY P. B. SCOTLAND, MORENCI, ARIZ.

(New York Meeting, February, 1915)

THE mines of the Arizona Copper Co. are situated in the Morenci-Metcalf copper district in southeastern Arizona. This copper-bearing district covers a triangular mountainous area of about 3 square miles, rising abruptly from the gravel plateau of the surrounding country. In this area, an immense intrusion of porphyry has displaced, shattered, and engulfed the shales, limestones, and quartzites formerly bedded on the basal granite.

Outcrops of oxide ore were discovered in 1873, and these surface ores were found to connect in depth with large bodies of high-grade oxide ores in the limestones and shales. The smelting of these ores in blast furnaces continued until about 1893, when the rapid exhaustion of the high-grade ore led to the discovery and exploitation of the low-grade sulphide ores in the adjoining porphyry.

Methods of concentration were then devised and the low-grade ores finally rendered profitable. The Arizona Copper Co. and the Detroit Copper Mining Co., operating in the same district, were the pioneers in the successful development of the low-grade "porphyry" properties of to-day.

The largest mines of the Arizona Copper Co. are the Humboldt and Clay at Morenci, and the Coronado, Metcalf, and King at Metcalf. These mines produce about 4,000 tons daily of concentrating ore containing from 2.5 to 2.8 per cent. copper.

The topography of the country is very rugged and mining operations are conducted principally through adit levels. A few of the mines are opened by shafts, the deepest being 1,100 ft., in the Coronado mine.

ORE OCCURRENCE

The low-grade sulphide ore occurs as: (1) disseminated deposits in the main stock of quartz porphyry; (2) lode deposits in fault fissures in granite. The high-grade oxide ores were found as tabular deposits in limestone and shale close to contacts with porphyry. Low-grade oxide ore formed the outcrop of some of the sulphide orebodies; the majority of the outcrops, however, were barren.

OREBODIES

Unlike most of the newer low-grade copper mines, the porphyry ore of this district does not occur as one large deposit. The main porphyry stock is broken up by the inclosure of large masses of sedimentary rocks resulting in a number of separate orebodies as dikes. Other orebodies have formed along fault-brecciated areas in porphyry or in granite, and the rock outside of the influence of the fault is barren or only weakly mineralized.

The diversity in size of some of the orebodies is shown by the following dimensions:

	Width Feet	Length Feet
Humboldt first orebody.....	80	600
Humboldt second orebody.....	200	700
West Petaluma orebody	6	450
Coronado orebody.....	35	1,750
King orebody.....	25	650

The depth from surface to which the orebodies have been proved to be of commercial value varies from 450 to 800 ft.

MINING METHODS

The method of extracting the ore is dependent upon the nature of the ore occurrence. Each orebody is a problem in itself. All influencing factors, as size, shape, position, nature and grade of an orebody, and character of the inclosing walls, are fully considered before the method of extraction is decided. The ore reserves, though large, are not so immense that mining methods involving a sacrifice of ore or its dilution by waste for the sake of cheaper mining costs have ever been used. The following examples are typical of the mining methods in use.

Opencast

The oxide ores found close to the surface on Metcalf Hill are extracted by opencast work. The amount of ore available for open work has never been sufficient to justify the large expenditures needed to install mechanical appliances such as steam shovels; hand work has always been employed.

Benches or terraces up to 30 ft. in height are carried into the orebody and the overburden and ore broken alternately. To prevent admixture of ore and cap rock, heavy blasts are seldom used.

The ore or waste is loaded by hand into cars of 25 cu. ft. capacity, or shoveled into chutes brought up from a lower level to the floor of the working place. In this handling of the ore, a preliminary sorting out of

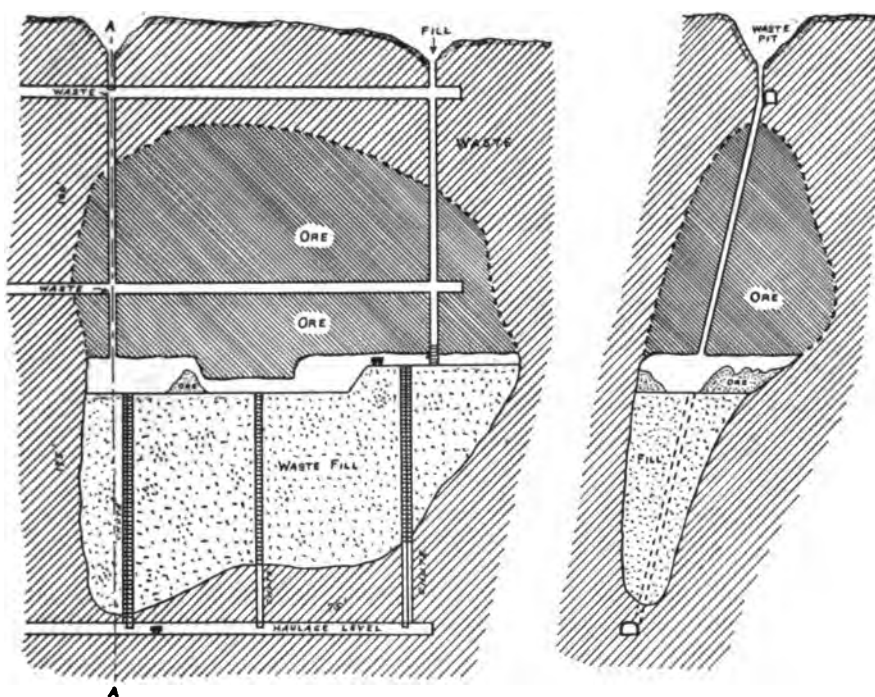
waste rock is made and this is completed on the sorting screens erected over the storage bins.

When conditions permit, mill-holing is used for the more rapid removal of the waste or ore.

The fluctuations in output of ore caused by the alternate handling of overburden and ore are reduced by keeping open a large number of working places.

If the orebody has been proved below the depth of economical open work, disposal of the overburden can be economically made by using it as filling material in stoping the ore on lower levels.

The dry climate and mild winters are favorable factors to opencast mining in Arizona.



Longitudinal Section.

Cross-Section A-A.

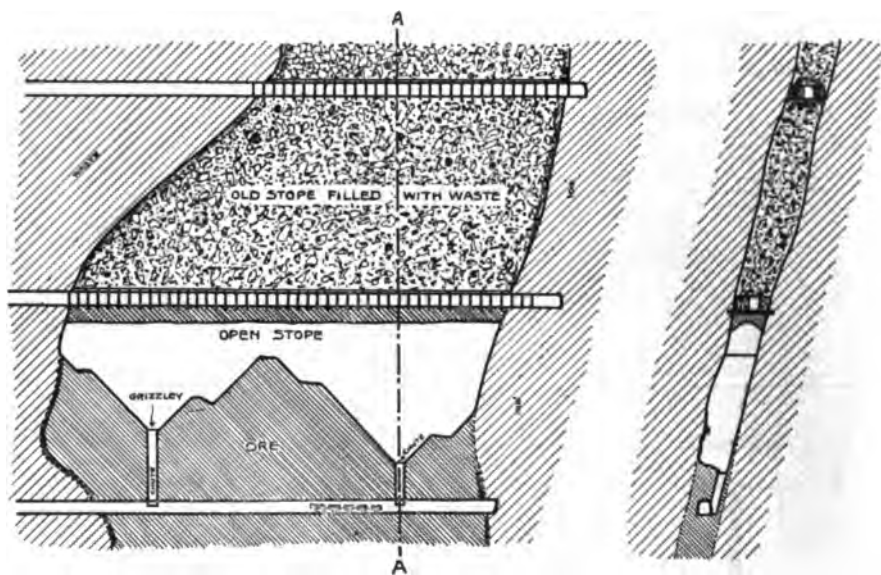
FIG. 1.—OPEN STOPING AND FILLING.

Open Stoping and Filling

Where the ore is hard and stands well, a system of open stoping without timber is employed with subsequent filling. (Fig. 1.)

The bottom of the oreshoot having been determined, it is opened for its full length and width and to a height of 15 to 20 ft. While this is proceeding, one or more raises are driven to the surface or to the level

above for ventilation and waste filling. Chutes and ladderways are then erected from the tramming level and the stope filled with waste to within 8 ft. of the roof. Slices of from 15 to 20 ft. in height are then mined overhand along the whole length of the oreshoot, and waste filling follows behind the working face. The filling is best distributed from the waste chute by means of a light car on a movable track. Mechanical means such as scrapers have not proved successful. Excepting a few log cribs or square sets placed beneath loose portions of the roof, no timber is used.



Longitudinal Section.

Cross-Section A-A.

FIG. 2.—UNDERHAND STOPPING.

Where the orebody is on or close to an important tramming road, stopping is started above the level of the road and a shell of ore 10 to 15 ft. thick is left to protect the roadway.

Several stopes in Metcalf mine worked by this method were so wide (up to 300 ft.) that pillars representing about 5 per cent. of the orebody were left to support the roof. If waste filling is cheaply obtained, either from surface stripping or from glory holes, the system is very economical, but the danger to the men from the unsupported roof limits this method to only the best standing ground.

Underhand Stopping

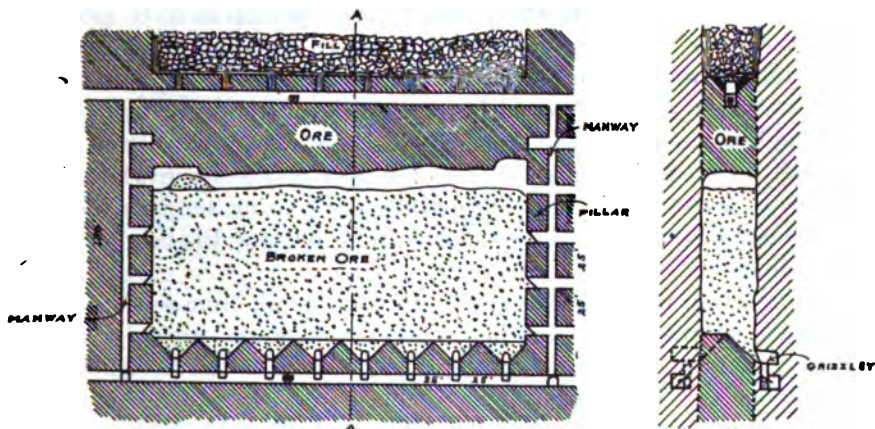
In narrow veins with firm walls, underhand stopping directly into chutes is employed. (Fig. 2.)

Untimbered chutes are carried to the top of the orebody at intervals of 25 to 50 ft. Around the top of these chutes underhand mining commences, the broken ore falling directly into the chute. The working floor is always kept cone shaped to avoid shoveling the broken ore. A grizzly of logs prevents very large pieces of ore from entering the chute.

This method is only practicable under a sound roof, as in the process of stoping the height of the roof is constantly increasing and its observation made more difficult.

Shrinkage Stoping

In this method, the broken ore is allowed to accumulate in the stope to support the walls and form a working floor for mining the roof. Practically no timber is used and the ore is drawn from chutes at the bottom of the stope directly into the mine cars.



Longitudinal Section.

Cross-Section A-A.

FIG. 3.—SHRINKAGE STOPPING.

In mining large orebodies shrinkage stoping is generally used in conjunction with top slicing or block caving; the pillars separating the shrinkage stopes necessitating the use of either of the foregoing methods for their removal.

The application of this method at the Coronado mine is shown in Fig. 3. The vein is about 35 ft. wide and is stoped in lengths of 150 ft. with pillars 30 ft. long between the stopes. The overhand mining is confined to the ends of the stope with the object of leaving a sag or belly in the roof of the ore. This finally breaks off by its own weight and, if it does not break by its fall, it is "bulldozed" on top. As stoping progresses, about one-third of the broken ore must be drawn out to allow working space below the roof. Should the roof become dangerous to work beneath, the ore can be broken down by underhand stoping from a

sub-level driven 25 to 30 ft. above the stope. Access to this sub-level is obtained from the ladderways in the end pillars.

The breaking of ore is stopped 25 to 30 ft. below the bottom of the overlying stope and the broken ore then rapidly drawn through the chutes. The old waste filling in the overlying stope is tapped and run into the empty stope. When full and leveled off, the shell of ore left in the roof is gained by square-set timbering. The pillars between the stopes are then removed by top slicing.

The chute arrangement at the bottom of the stope varies in different orebodies. At the King mine, the ore breaks fine and can therefore be pulled directly through the chutes. At the Coronado mine, the ore is harder and cannot be run directly into cars without much block-holing in the chutes. Two different methods are used to allow of the ore being broken by hand. In the first system, haulage roads are run in the foot wall parallel with the orebody and inclined chutes put up at 30-ft. intervals into the stope. Over each chute is a grizzly on which the laborer stands to rake down the ore and break the large pieces passing into the chute. Another system employed is to run a level, timbered two sets high, through the center of the orebody. On the sides of the upper set of this drift inclined raises are made to the floor of the stope which is opened 20 ft. above the level. The broken ore passing through these raises is loaded directly into the cars beneath by opening the lagging on top of the first set.

The completion of a stope depends upon its depth from surface or the condition of the upper workings. If on top of the orebody, the stope can generally be carried to cap rock and then drawn quickly without difficulty. The danger of an air blast from the sudden caving of an empty stope makes subsequent filling always advisable.

Where the roof of the stope is too weak to leave an unsupported shell, the roof is carefully arched and timbered with long heavy stulls from wall to wall, before the broken ore is drawn.

A method was tried of laying a heavy mat of timbers on top of the broken ore, caving the waste above on top of this mat and drawing the ore from beneath. It was found that if the mat was caused to descend more than 30 ft. it became so broken up that a serious dilution of the ore with waste resulted.

A vital objection to this system developed in the Coronado mine. The ore from this mine contains a larger amount of pyrite than is usual in the district, and when the broken ore was drawn from the stopes it was found to be so highly oxidized that the system had to be abandoned.

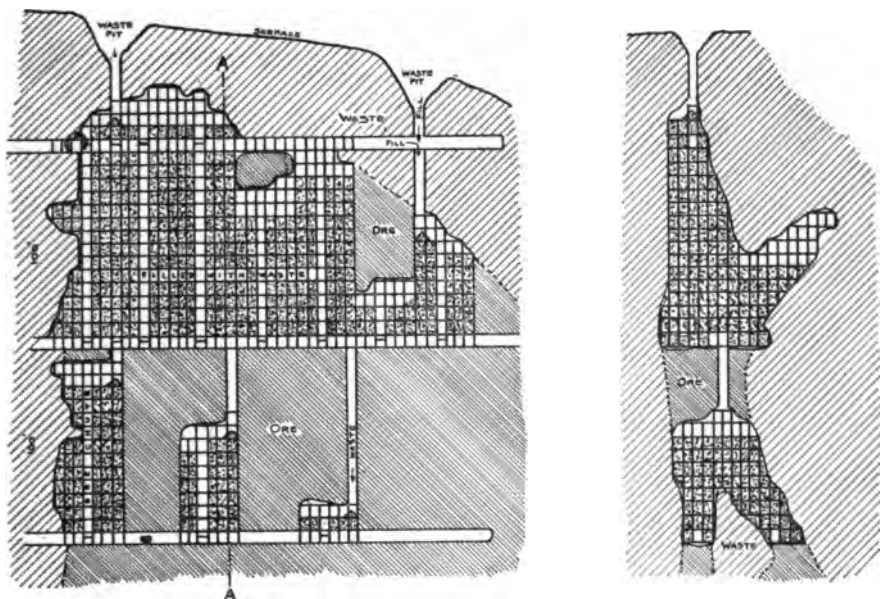
Square Set and Fill

The principal mining method employed for many years in the district was square set and fill. (Fig. 4.)

The expense of using this system in the large bodies led to its disuse and the substitution of top-slicing methods. An excessive amount of timber had to be used in reinforcing the square sets to prevent serious caves, and the maintenance of raises to surface for waste became more difficult as the workings extended.

The square-set method is, however, still employed in the extraction of orebodies of irregular shape and in the first stage of top-slicing. Except in minor details, square-set practice here does not differ from that of other districts.

The set of timber comprises an 8 by 8-in. cap, 6 by 8-in. brace, and a 6½-ft. round post. The set allows of the extraction of 178 cu. ft., or



Longitudinal Section.

Cross-Section A-A.

FIG. 4.—SQUARE-SET METHOD.

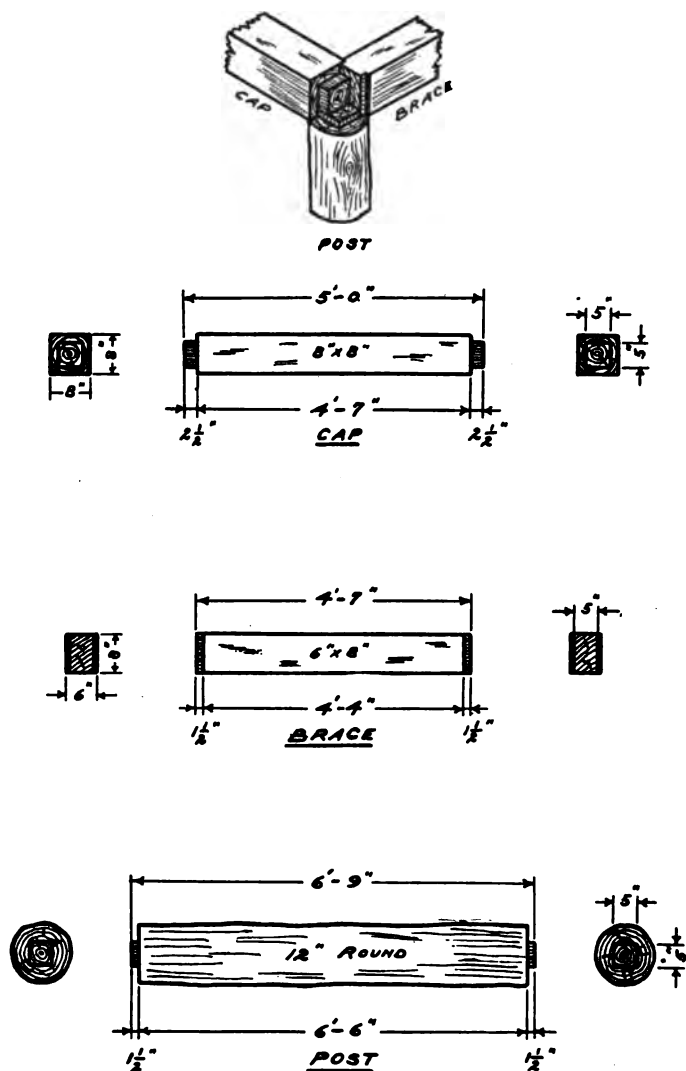
14 tons, of ore, and without any reinforcements calls for 8 ft. b. m. of timber per ton of ore. (Fig. 5.)

Waste filling is obtained from raises to surface or to old stopes above and the floors are kept filled to within one set of the roof.

As the high cost of timbering in square-set work is due to the necessity for reinforcing caps and braces that are overloaded, stopes of small floor area are opened and worked rapidly.

A change in the square-set practice has recently been made by using sets 10 ft. high, allowing of the extraction of 250 cu. ft. of ore. The stope posts of 6½ ft. height have been replaced by posts 9½ ft. long; the other set members have not been changed.

The increase in height of the set causes only a small reduction in its strength and reduces the cost of timbering about 15 per cent. In a stope, the weight of the roof is carried by the caps, braces, and lagging and through these transmitted to the posts. The crushing strength of



Sets are 5 ft. square and 7 ft. 2 in. high, centers.

FIG. 5.—SQUARE-SET FRAMING.

the $6\frac{1}{2}$ -ft. posts is so much greater than the transverse strength of the cap and brace that an increase in the height of the post does not materially reduce the strength of the set.

Sub-Level Caving

A sub-level caving system is sometimes employed in mining the upper part of an orebody preparatory to top slicing.

The details of the system are shown in Fig. 6. Slices 40 ft. wide are cut up by drifts 20 ft. below the caved ground. The back of ore is mined by retreating and the waste allowed to cave. As no mat is used, the inevitable mixing of ore with waste rock causes a serious fall in the grade of the ore.

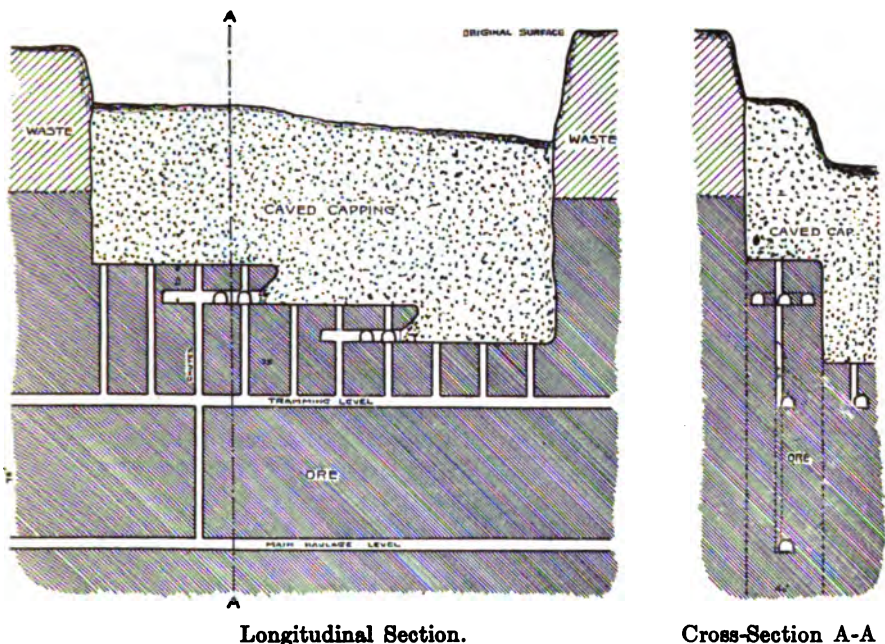


FIG. 6.—SUB-LEVEL CAVING.

Rill Stopping

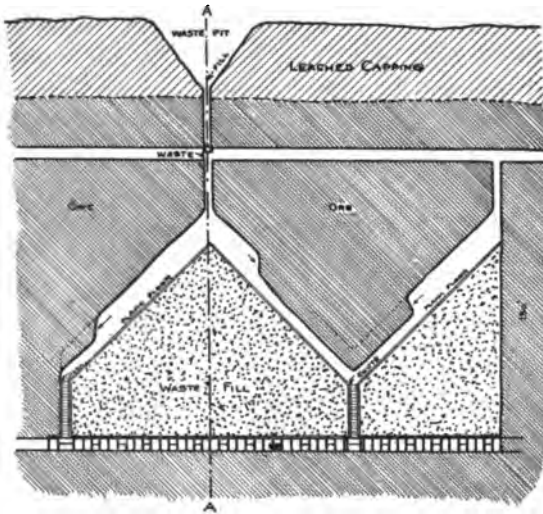
In narrow veins where filling is needed to support the walls, rill stopping is often employed. (Fig. 7.)

In this system the roof of the stope is worked on an incline and the broken ore runs down the sloping fill directly into the chute. After a slice of ore 10 to 15 ft. in height has been worked out, the stope is cleaned, waste filling run in and mining resumed.

Shoveling is eliminated, but the output of ore is necessarily intermittent.

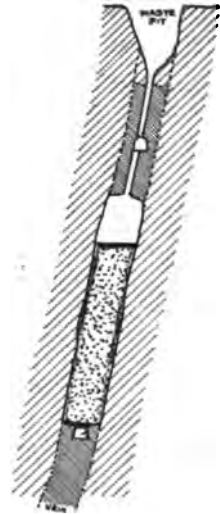
Top Slicing

The top-slicing method is principally used in the extraction of the large, soft ore orebodies of the Humboldt mine. The system is an appli-

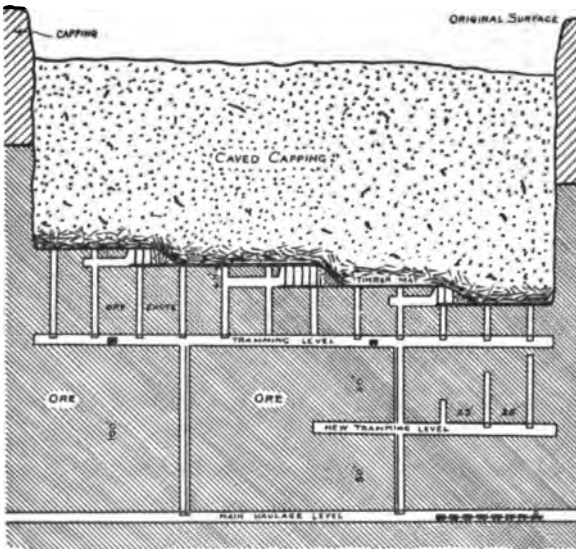


Longitudinal Section.

FIG. 7.—“RILL” STOPING.

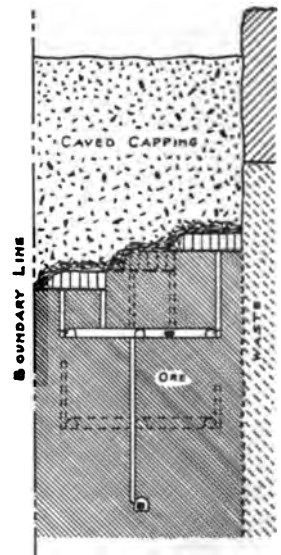


Cross-Section A-A.



Longitudinal Section.

FIG. 8.—TOP SLICING.



End Section.

cation of longwall retreating of the coal mines, worked under an artificial roof of timber called the “mat.” (Fig. 8.)

The top of the orebody is removed either by open stoping, square setting, or sub-level caving and a double floor of 2-in. plank laid. The overburden is then caved on top of the mat, either to surface or sufficiently high to make it safe to work beneath the mat.

A tramming level is laid out 55 ft. below the mat and divided by drifts into blocks of 30 to 40 ft. in width. At intervals of 25 ft. along these drifts, timbered raises divided into chutes and ladderways are carried up to the mat above. Commencing at the boundary of the ore, slices 11 ft. in height by 30 or 40 ft. in width are mined beneath the mat, which is temporarily supported by sets with posts 10 ft. high and round unframed caps. As the slice advances, a layer of 2-in. plank is laid on the floor, lengthwise with the slice, and the timber mat is caved by blasting the posts behind the working face; only sufficient space is kept open under the mat to permit of shoveling into the chute. Connection is made between the chutes by driving a tunnel ahead of the slice. This tunnel may be small and untimbered or carried the full height of the slice and timbered with 10-ft. sets to support the mat.

From the drift chutes, the ore is transferred by hand tramming in cars of 21 cu. ft. capacity to the motor chutes on the main haulage roads.

Top slicing allows of a more regular and larger output from a working breast than square setting. The breaking and shoveling of the ore is, however, not so advantageous as in square sets; the working face not being undercut and the shoveling being done on a rough floor. The weight of mat is heavy but fairly constant and sudden caves are exceptional. Owing to the smaller working space that must be kept open, the consumption of timber is less than with square sets.

To allow the broken ore at all times to be shoveled directly into the chutes, the following modification has been made: When the working face has advanced so far that the ore cannot be shoveled directly into the chute, the men are transferred to the next chute, and a new breast is opened and worked on all sides. In slices 30 ft. wide and with chutes 25 ft. apart, this system always permits direct shoveling and avoids the use of wheelbarrows.

Panel Slicing

To eliminate the heavy expense of installing and maintaining chute raises every 25 ft. in the slices, a system whereby the ore will be loaded into cars directly in the slice is about to be tried out. In this system, which has been called "panel slicing," all of the working faces and all tramming will be on the same floor and the use of tramming levels at 55-ft. intervals will be abandoned. (Figs. 9, 10, and 11.)

The working level will be divided into slices by crosscuts every 40 ft. along the main road. The main road and the parts of the crosscuts

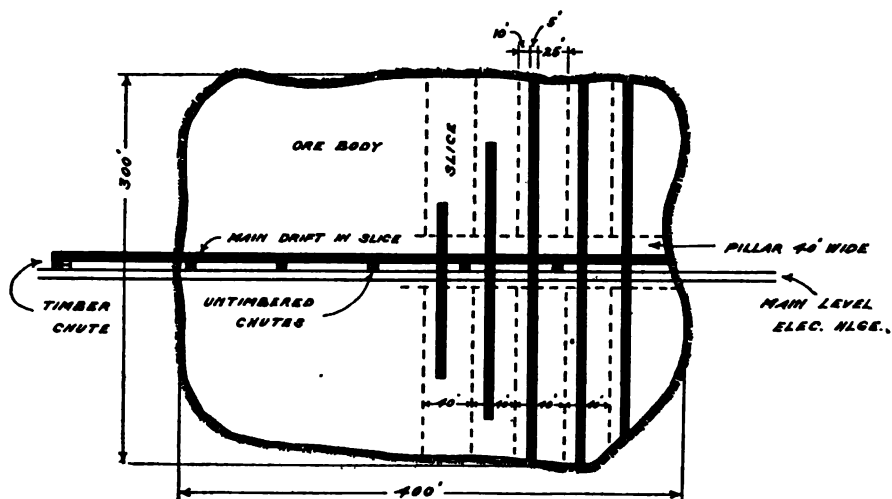


FIG. 9.—PANEL SLICING. PLAN.

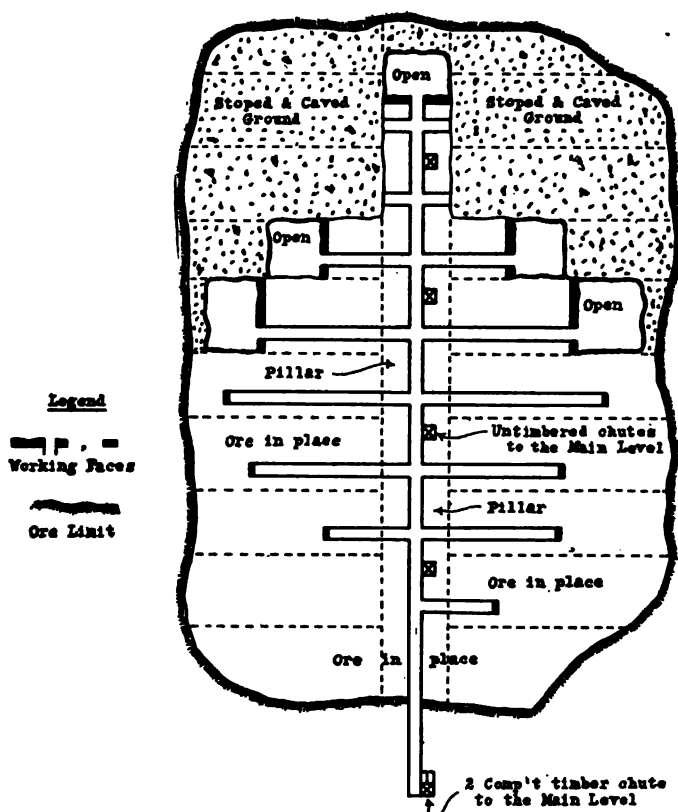


FIG. 10.—PANEL SLICING. PLAN OF WORKING FACES.

forming the main road pillar 40 ft. wide will be timbered with sets $6\frac{1}{2}$ ft. high. Outside of this area, the crosscuts will be opened through to the mat above and timbered with slice sets 10 ft. high. Slices 40 ft. in width will then be worked from the margin of the ore toward the pillar over the roadway. This pillar will not be removed until the slices on each side of it are exhausted.

Ore chutes spaced at 80-ft. intervals along the main level receive the ore trammed from the working faces.

As one floor or panel is being worked, the next one, 11 ft. below, will be in preparation.

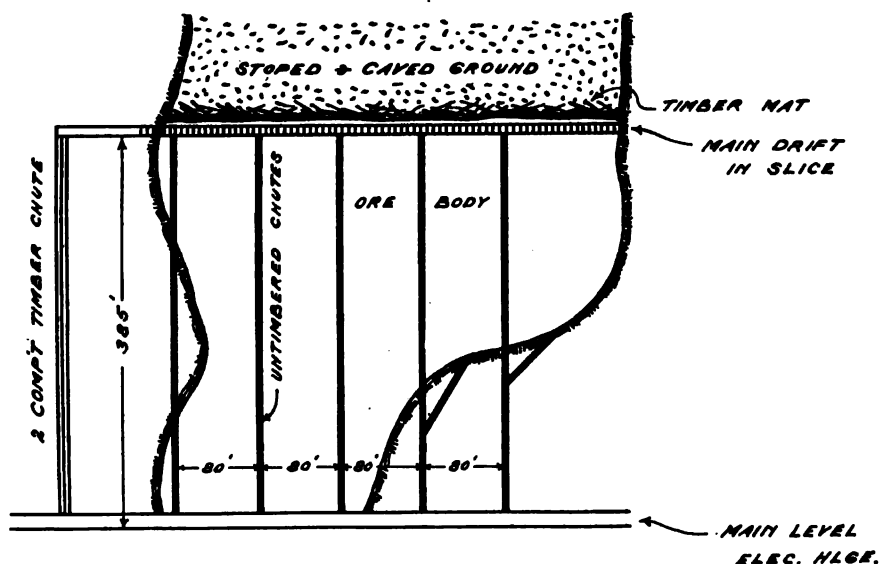


FIG. 11.—PANEL SLICING. ELEVATION.

Drilling and Blasting

Air drills are employed almost entirely in mining. Air at 100 lb. pressure is supplied by two Nordberg compressors of 4,000 cu. ft. total capacity, direct connected to two alternating-current motors. One compressor at Morenci supplies the Humboldt and Clay mines; the other at Coronado supplies the King and Coronado mines.

One-man machines of various makes are used throughout. In slicing, "jackhammers" are used in soft and water Leyners in harder ground.

Giant ammonia powder of 30 to 40 per cent. is used in blasting, 1-lb. of powder breaking from 7 to 10 tons of ore. Prepared primers are issued to the miners from a number of supply magazines in the mine.

Timber and Timbering

Oregon fir costing about \$24 per 1,000 ft. is used for all square timbers, round timbers being of Texas pine costing about \$19. Round timber, being stronger and cheaper, is used wherever possible. In temporary timbering, such as in top slices, cheap cull lumber is used.

The standard sizes of timber sets are cut by a double-end framer at the terminus of the standard-gauge railroad at Clifton, before shipment to the mines.

Holman air hoists are used in handling the timbers to the stopes.

The dimensions of the timbers employed are as follows:

	Length feet	Diameter inches,
Top-slice posts.....	10	7½ to 9
Top-slice caps.....	10	7½ to 9½
Top-slice flooring.....	10-15	2 by 12
Square-set posts.....	$\left\{ \begin{array}{l} 6\frac{1}{4} \\ 9\frac{1}{4} \end{array} \right.$	$\left\{ \begin{array}{l} 8 \text{ to } 12 \\ 8 \text{ to } 12 \end{array} \right.$
Square-set caps.....	5	8 by 8
Square-set braces.....	4¾	6 by 8
Tunnel posts.....	6	10 to 12
Tunnel caps.....	5	11 to 13
Motor tunnel posts.....	7½	10 to 14
Motor tunnel caps.....	7½	12 to 16

The consumption of timber in top slicing, including chutes, ladders, and reinforcements, is about 9 ft. b.m. per ton and in square-set work about 15 ft. b.m. per ton.

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The Mining and Reduction of Quicksilver Ore at the Oceanic Mine, Cambria, Cal.

BY C. A. HEBERLEIN, LOS ANGELES, CAL.

(New York Meeting, February, 1915)

INTRODUCTION

THE present war in Europe seems to have stimulated the demand for quicksilver. In July last, the price ranged around \$35 per flask of 75 lb., while to-day it seems to fluctuate between \$47.50 and \$50. The use of quicksilver in the form of fulminate, containing 70 per cent., and as corrosive sublimate, containing 73.8 per cent., must be extensive and a further market advance may be expected. With such an advance, some revival in quicksilver mining may be looked for, particularly in California, where many abandoned properties exist. What can be done with low-grade quicksilver ores under favorable conditions, particularly with low mining cost, may be seen from the following description of the Oceanic mine, near San Luis Obispo, Cal. Its operations date back to 1876, when rich sand ores were discovered. In three years 7,400 flasks of quicksilver were produced, but since then the deposit has been only sporadically worked.

Until two years ago various attempts had been made by the Oceanic Quicksilver Co. to mine the more refractory "mud rock," which was encountered lying beyond the "sand rock." Mud and sand rock in varying quantities were treated together with poor results, but operations were successful with sand rock of fair percentage. About two years ago, when all the sand rock had been exhausted, Murray Innes purchased the Oceanic mine. He found large quantities of low-grade mud rock, varying from 6 to 8 lb. of quicksilver per ton, which at the prevailing low price of quicksilver demanded very cheap mining costs, in conjunction with high but cheap metallurgical extraction. At that time the extraction of quicksilver from mud rock ranged between 40 and 50 per cent., as far as I am able to learn.

MINING

The low cost of mining was the principal consideration, since, with mercury at 50c. per pound, the total gross value of the ore did not exceed

\$3 to \$4 per ton. Square setting and filling was out of the question, so various attempts with cheap methods were made, among them a shrinkage system. None gave satisfaction on account of the peculiar character of the ground. The mud rock being soft and slaky became easily dangerous and impossible to work.

The pay vein averages 20 ft. in width, and has its own local character-

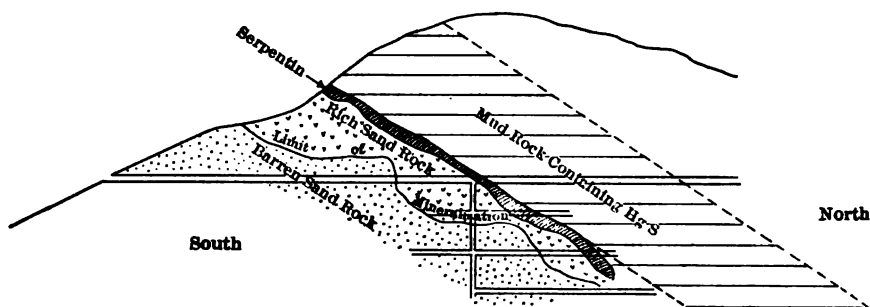


FIG. 1.—LONGITUDINAL SECTION OF THE OCEANIC MINE.

istics. The foot wall stands nearly vertical and is either serpentine or barren sand rock. The vein proper is a contact vein of typical "mud rock." The first 10 or 12 ft., as a rule, is coarse, hard mud rock and contains cinnabar only; then comes from 7 to 10 ft. of fine, darker mud rock containing both native quicksilver and cinnabar; then follows a gradual transition of vein matter into barren mud rock. (See Fig. 1.)

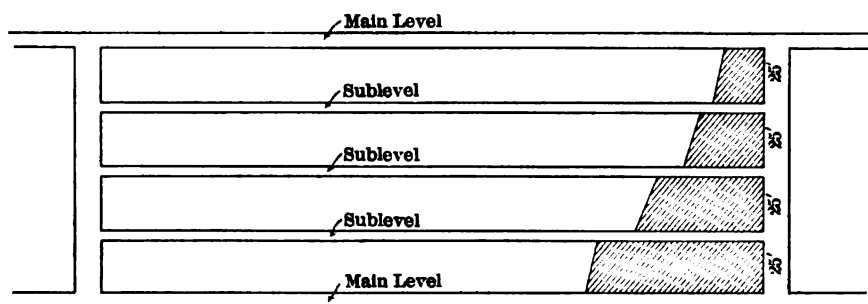


FIG. 2.—SUBLEVEL SLICING SYSTEM OF MINING.

The method of mining locally employed is as follows: The whole ore-body between two main levels 100 ft. vertically apart, for a longitudinal distance of 300 or 400 ft., is attacked by two raises, which are connected by intermediate levels 25 ft. apart. (See Fig. 2.) Long holes are drilled with hammer drills from below and above on one end of the slice and shot. The slicing proceeds from one end of the block, and the ore is trammed to

the other raise. Shooting from above and below at the same time, the intervening slice is completely broken up. What broken ore falls into the drifts is dumped into the chute; most of the ore is handled from the level below.

The cost of driving in the mud rock with hammer drills ranges from \$1 to \$2 per foot; and as the driving is done entirely in ore, the mining of the latter costs only from \$1 to \$2 per ton. Locally this has proved the most efficient and cheapest method that can be practiced without endangering the lives of the miners. If large pieces should come down they are easily broken up from below. The ground is ideal for hammer drills, which are used both in stoping and drifting.

REDUCTION

Two methods are employed for the treatment of the ore. Wet ores are first concentrated and then treated in the Scott furnace; but the dry ores are directly charged into the Scott furnace of the usual type.

The ore is crushed at the mine to about $1\frac{1}{2}$ in. in diameter and brought to the furnace plant by a bucket tramway. The wet ore is fed by a Challenge feeder into a $3\frac{1}{2}$ -ft. Huntington mill provided with 16-mesh screen. Without settling the undersize passes directly to a Deister table.

Since the gangue is more friable than the ore, a Huntington mill was selected for fine grinding; and little sliming takes place. The charge contains about 0.3 per cent. Hg, or 6 lb. per ton, and the concentrates carry about 5 per cent. Hg, or 100 lb. per ton. From present results, the extraction seems to be 80 per cent. and the concentration ratio 20 into 1. It requires three men for attendance at \$7.50 per 24 hr. and \$2.50 for power, or a total of \$10 for a capacity of 20 tons, or 50c. per ton. The concentration of the wet ore is successful at the Oceanic but more costly than direct furnace treatment. The concentrates are dried and then charged into the Scott furnace, but not into the retort, as this would prove too expensive.

Some experiments have been made lately with the flotation process on quicksilver ores, but it is hardly probable that this process ever will find application, for two good reasons: (1) The fine grinding alone would cost as much as the ordinary furnace process; and (2) the oil sticking to the concentrates would distill over in the retort and severely impair the quicksilver, which would have to be specially cleansed of its coating of oil.

The furnace plant at the Oceanic consists of a 50-ton Scott tile furnace, one brick condenser, and three wooden condensers. In the last two years' operations many improvements have been made, among which are the automatic sealing of the charge; the reinforcing of the tiles; and radical changes in the firing and the system of condensing.

Charging.—The old method provided for shutting off the ore a flat

cast-iron gate, which caused endless trouble. The gases, being very acid, attacked the iron of the gate, which would not last at times more than two months, and its destruction would mean a shut-down. The escaping gases are very harmful to the men charging, besides being the source of considerable loss. Mr. Innes designed a simple and inexpensive arrangement, which served all purposes (see Fig. 3). Above the long charging

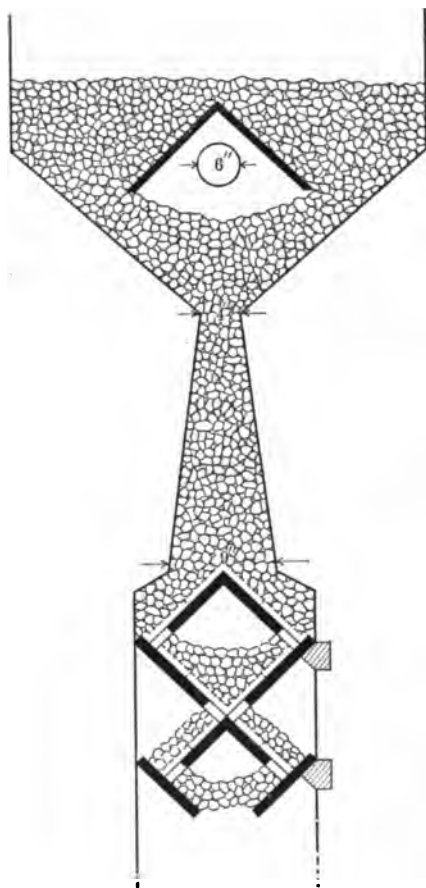


FIG. 3.—AUTOMATIC SEAL OF CHARGE PIT ON FURNACES TREATING MERCURY ORES.

chute (which measures 9 in. at the bottom, above the first tiles, and 4 in. at the top), he placed a cast-iron spreader to trap the escaping gases, which are drawn off by a 6-in. cast-iron pipe into a special water-cooled wooden condenser. The spreader plates are made of cast iron, 18 in. by $1\frac{1}{2}$ in. by 4 ft. long, and lie loose on an iron saddle, so that in case a bar has to be used, they can be turned back, giving full access to the throat. The opening on either side of the spreader is 6 in., but enough ore is charged to

keep the spreader covered all the time. The sides of the spreader must be kept covered high enough to prevent the escape of gases. As the ore, after being burned sufficiently, is discharged below, fresh ore follows from above automatically. This method prevents sudden rushes and the even draft of the furnace is never disturbed—an ideal condition for good furnace work.

Reinforcement of Tiling.—The Scott furnace, tiled and reinforced as described, has run for two years and has never given the slightest trouble. Not only are the tiles supported by fire brick, but also their end joints are covered with fire brick, which has a strong and steadying effect in case ore should be hung up and started again with the tickler through the pigeonholes. The tickler is a long iron rod with a jointed end, folding back in a certain position to start the ore again by moving it backward and forward. The tiles in a quicksilver furnace have to be handled with the utmost care, since, in case a lower tile should loosen and give away, the upper ones are apt to follow, causing a general shut-down of the furnace. The shutting-down of a Scott furnace is not like that of a smelting furnace, a question of hours or a day, but one of weeks, a very serious matter since production ceases entirely during this time. Hence the above details are very important to a practical furnace man and should be borne in mind.

Change in Firing.—The system of roasting the ore has been changed to one of sublimation, or rather one of handling the furnace on the principle of a retort.

On opposite sides of the Scott furnace were fire boxes for burning cord wood on grate bars. The latter have been entirely removed and the fire boxes bricked up solid, just enough space being left on the extreme end for ashes to drop into the ash pit below. Ash pit and fire box are tightly closed; but enough air seems to enter through cracks to keep the wood in a state of glow. Formerly $2\frac{1}{2}$ to 3 cords of wood were burned in 24 hr. on the same ore as at present, with a production of from 40 to 50 flasks of quicksilver. Now, less than $\frac{1}{2}$ cord of wood produces not less than 100 flasks per month on a daily treatment of 50 tons of ore. These extraordinary results are easily explained. The mud rock contains considerable marcasite (FS_2); and it was a great mistake to roast the ore until it had been burnt red. Plenty of air entered through the fire box and the pigeonholes, causing a much higher heat than was necessary, and also, by roasting all the marcasite, producing a large amount of SO_2 and SO_3 gases. The heat and draft made large amounts of soot of low value, keeping the retort going nearly all the month. At the present time the soot is cleaned up in a few days, and metallic quicksilver is recovered, which was lost before. No air enters the furnace except what leaks in at the bottom and through cracks in the fire box. The bottom of the fire box is sealed tightly, as are all pigeonholes. The tem-

perature in the furnace is a dull red glow, and most of the time the burned ore is black, instead of red, as formerly. Pannings show no cinnabar; and the sublimation of the mercury seems to be absolutely complete. The assay of the ore, all the year around, does not seem to average more than from 6 to 7 lb., and assays I have made by means of the distillation method ran from 6 to 6½ lb. of quicksilver per ton. Assuming 6½ lb. as the average, the contents in a month's run would be:

$$\frac{1,500 \text{ tons} \times 6.5 \text{ lb.}}{75 \text{ lb. per flask}} = 130 \text{ flasks per month.}$$

The average monthly output is about 110 flasks, and allowing for 8 flasks retained in the dust chamber, which is recovered only when the furnace is shut down, the actual extraction is

$$\frac{118}{130} = 90.8 \text{ per cent.}$$

This is an excellent extraction with the present condenser plant, and is principally due to the radical metallurgical change pointed out above.

The assay of ore so low in grade as the Oceanic offers considerable difficulties; and a chemist not familiar with mercury ores will have some trouble in checking results. Locally, the pan is generally used as the most expedient means, and with some experience differences in the grade of the ore are readily distinguished. This seems a crude way but it proves to be as accurate as is necessary. After having hundreds of samples run, there seems to be little difference in the grade from the different levels. The ore is pretty well disseminated in fine grains all through the rock, which looks like waste, very little cinnabar being visible to the naked eye. After moistening the rock little dots of ore may be seen here and there; but the mass hardly looks as rich as it actually is (carrying from 0.3 to 0.4 per cent. of quicksilver). The cinnabar is always bright red; but the native quicksilver is not always easily visible, particularly when the ore has been mined some time. The proportion of native to cinnabar is not known but on the whole the former is of minor interest.

Condenser Plant.—The condensing system has also undergone an entire change during the last year. Formerly all condensers were made of brick, as was only natural, in view of the excessive heat developed. But brick is not a good material for condensers, being poorly adapted to withstand the corrosion of the gases and the use of water during clean-ups—the latter hastening materially the deterioration of the brick.

By the time the furnace gases leave the cast-iron downtake and enter the fire-brick condenser, the temperature is hardly above 250° C. They enter the brick condenser at the top and are drawn off at the bottom, where the temperature has dropped to 170° C., and are then taken into

the first wooden condenser, made of tongued and grooved redwood. This soon sweats out the pitch and seals itself tight, appearing as if it had been painted. Formerly divisions were built in the condensers and the gases were let in at the bottom to be circulated up and down; but this has been found less efficient than letting in the gases at the top and drawing off at the bottom. As long as the gases are warm, they have a tendency to rise, hence by drawing off at the bottom the coolest gas is taken. The tops of the wooden condensers are easily water-cooled. The condensers are square wooden boxes of the simplest construction, with a good, tight bottom. The gases have in them a chance to expand, cool through radiation, and gradually descend. This principle is followed through the whole condenser system. The redwood seems to radiate much better than brick, and easily withstands a temperature of 170° C.; so there is no reason for the use of brick, which is more expensive and less efficient. What brick condensers still remain at the Oceanic mine will be replaced by wooden ones as soon as the change can be made.

Many experiments have been made with all kinds of baffling, but without any success. A minimum amount of draft and maximum amount of expansion gives the best results. All fanciful apparatus with water-cooling or air-cooling pipes, etc., is simply worthless and expensive. The flow of gases must be a slow one and all possible room for expansion must be given. No acid-proof mortar has been found to withstand the corrosion of the gases; but the redwood seems to fill all requirements. From the second (redwood) condenser, about 1,000 gal. of water run out daily. This is the condensed moisture of the ore. It is very acid and carries quicksilver in solution, which will be precipitated in future and may yield as much as a flask per month. I have no doubt that the extraction of quicksilver from this low-grade mud rock will eventually be brought up to 95 per cent., which certainly is a decided progress, when we consider that a few years ago the extraction hardly reached 50 per cent.

COSTS AND PROFITS

Since there is likely to be an increased demand for quicksilver in the future, the question naturally arises, What grade of ore can be profitably worked?

The size of ore deposit, cost of mining, wages, and fuel ought to be carefully considered before making the first outlay, which in the mining and reduction of quicksilver is not a trifling matter. An efficient 50-ton plant, with all accessories, will easily cost \$25,000 or more; and there should be a considerable tonnage of ore in sight to warrant such an outlay of capital.

At the Oceanic mine the cost of mining is exceedingly low, for the reason that the vein is soft and easily stoped. Wages are low also.

Miners are glad to work for \$2.50 per 8-hr. shift, on account of the low cost of living in a dairy and agricultural country. Wood costs \$5 per cord, delivered at the mine.

The reduction costs for all furnace operations on 50 tons of ore daily are as follows:

Three charge men	@	\$2.50		\$7.50
Three furnace men	@	2.50		7.50
One helper	@	2.50		2.50
One foreman	@	3.50		3.50
One-half cord of wood	@	5.00		2.50
Total on 50 tons				\$23.50
Per ton of ore				0.47

Adding the handling of the soot in the retort, the total cost is about 50c. per ton of ore.

With a recovery of 91 per cent., or 5.91 lb. of quicksilver per ton, worth 50c. per pound, the value recovered would be \$2.95 per ton, and the total cost would be (\$1 for mining and 50c. for reduction) \$1.50, leaving as net profit per ton of ore \$1.45. This is but a small margin of profit, and it is evident that at the figures named, close management and a large tonnage of ore are necessary to make a successful business.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Testing and Application of Hammer Drills

BY BENJAMIN F. TILLSON, FRANKLIN FUERNACE, N. J.

(New York Meeting, February, 1915)

THE hammer drill rightly receives the credit for having made the one-man drill possible, and so many economies seem possible through the proper application of different types of hammer drills to various mining, quarrying, and excavating operations, that an indication of the economies effected by the New Jersey Zinc Co. at its Franklin mines may be of pertinent interest. When this company commenced its trials of hammer drills in 1907, these tools had not been developed to one-fourth the capacity and refinement which they have at present. At that time it was frequently stated that such a small tool, drilling holes of less diameter than the reciprocating rock drill, could not drill enough holes in a shift to permit the placing of sufficient explosive to break a tonnage of ore comparable with that produced by the "rock drills;" that the placing of small holes inclined upward, at angles steeper than 40° above the horizontal, could not be expected to produce results equal to the large flat, wet or dry, holes in the breasted back of an over-hand stope, and would only shatter the ground so as to make the back unsafe. In spite of these adverse opinions, the hammer drills first showed their superiority over both heavy and light reciprocating drills in raising and in stoping, and then in drifting and quarry work. As a result, all of the reciprocating drills at the Franklin mines were scrapped three years ago, all of the mining work being accomplished with increased efficiency, as shown in detail in this article.

With the advent of the hammer drill in this property, it was considered advisable to make comparative tests of all the tools accessible, and it has since been the policy to investigate the merits of any advance of the drilling art in order to get the maximum amount of work from the tools. The necessity of devising some means of standardizing drill tests, and of measuring the consumption of compressed air as well as the drilling speed, was early realized.

The common test was to fill a measured air receiver with compressed air at a certain gauge pressure, run the drill until the pressure had dropped to too low a figure, then compute from the time, drop in pressure, and

capacity of the receiver, the cubic feet of free air used. This was not considered a fair indication of the drilling capacity of a machine, since the performance of some drills did not vary directly with the absolute pressures of the compressed air.

It was, therefore, found expedient to build a water-displacement air meter with which the drill test could be carried on for any length of time without serious variations in the desired air pressure. This apparatus, as shown in Fig. 1, consists of two tanks, half-filled with water, and made of 12-in. pipe with blank flanges, gauge glasses being mounted on one, a four-way cock connecting the compressed-air supply pipe with both tanks and the air line going to the drill. This device gives more accurate results than the common types of water or gas meters; and since any errors are due to the human element of reading the gauge glasses and reversing the four-way cock, they tend to be compensating throughout a number of tests.

The procedure is as follows: Air is drawn by the drill from the receiver *C*, which tends to trap any moisture carried over by the air from tanks *A* and *B* and assures a constant pressure while the four-way cock is reversed. In the arrangement shown in Fig. 1, the receiver draws its supply of air from the tank *B* and the water rises in this tank by virtue of the pressure of the air admitted from the air main through the four-way cock to the top of tank *A*, where the water is being forced downward and through the 2-in. connecting pipe to tank *B*. When the water has risen to a certain point near the top of the gauge glass in tank *B*, the four-way cock is reversed and the inlet air is supplied to the top of tank *B*; the drilling air is then taken from the top of *A*, the reversal of the cock again being made when the water in tank *B* has fallen to a point near the bottom of the gauge glass.

A pet cock is placed on the top of tank *C* so as to permit the bleeding of air to bring the water columns to the desired point for starting a run, and another pet cock is attached to the bottom of the same tank in order to permit the drainage of water. For convenience in measuring and computing, a run is made on the supply of compressed air indicated by a certain number of reciprocations of the water columns between fixed points on the gauge glasses; the pressures are measured on the air gauge mounted on tank *C*, the length of time is taken by a stop watch, and the consumption of free air per minute is computed. Unless a pressure regulator is installed between the four-way cock and the receiver *C*, or else a globe valve at this point is operated manually to throttle the air so as to maintain a constant pressure, it is evident that the air pressure at the drill will vary in accordance with the water column supported by the inlet air pressure, but since the gauge-glass marks are in this instance set $34\frac{3}{4}$ in. apart, the maximum variation in pressures is about $1\frac{1}{4}$ lb. per square inch; it is difficult to find a pressure regulator which will control a pressure of 90 or 100 lb. per square inch to a closer

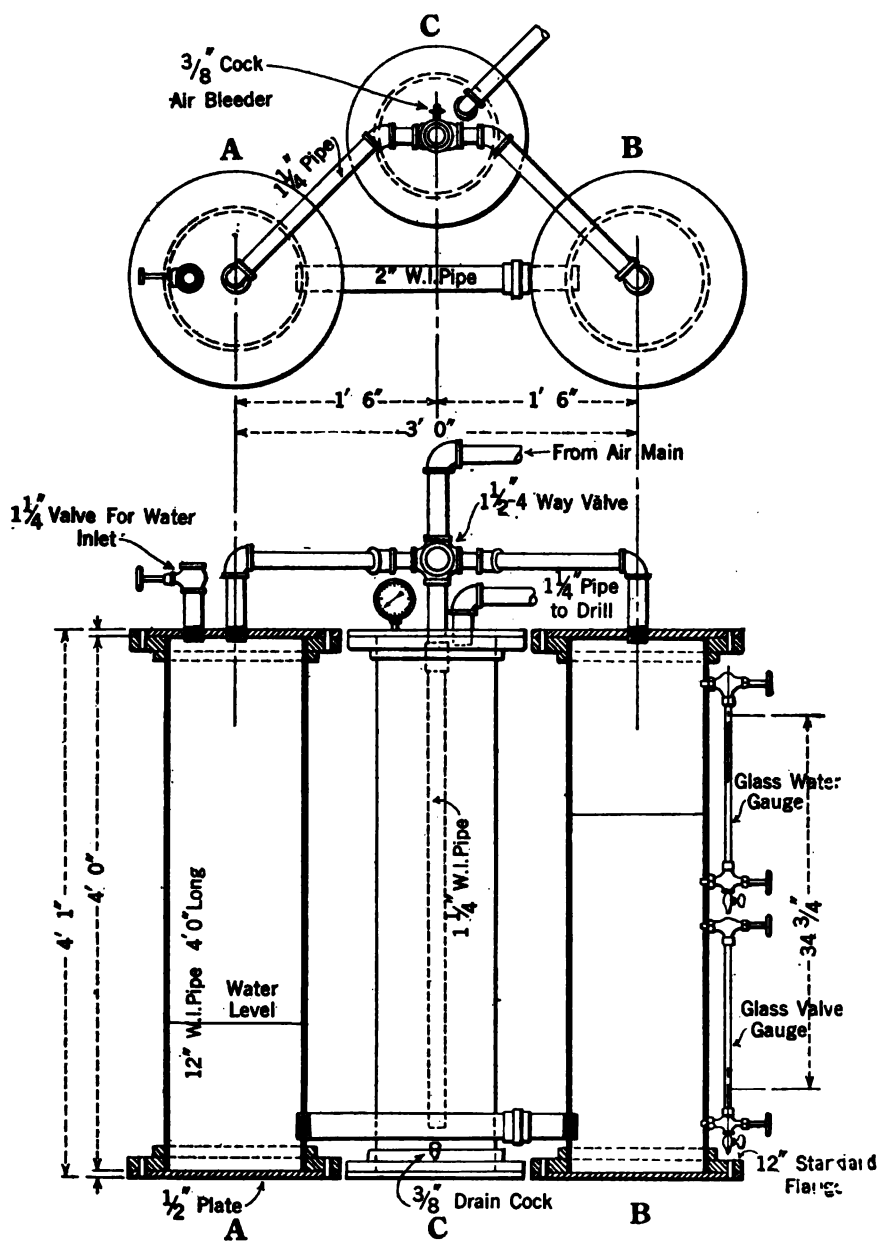


FIG. 1.—WATER DISPLACEMENT AIR METER FOR ROCK-DRILL TESTING.

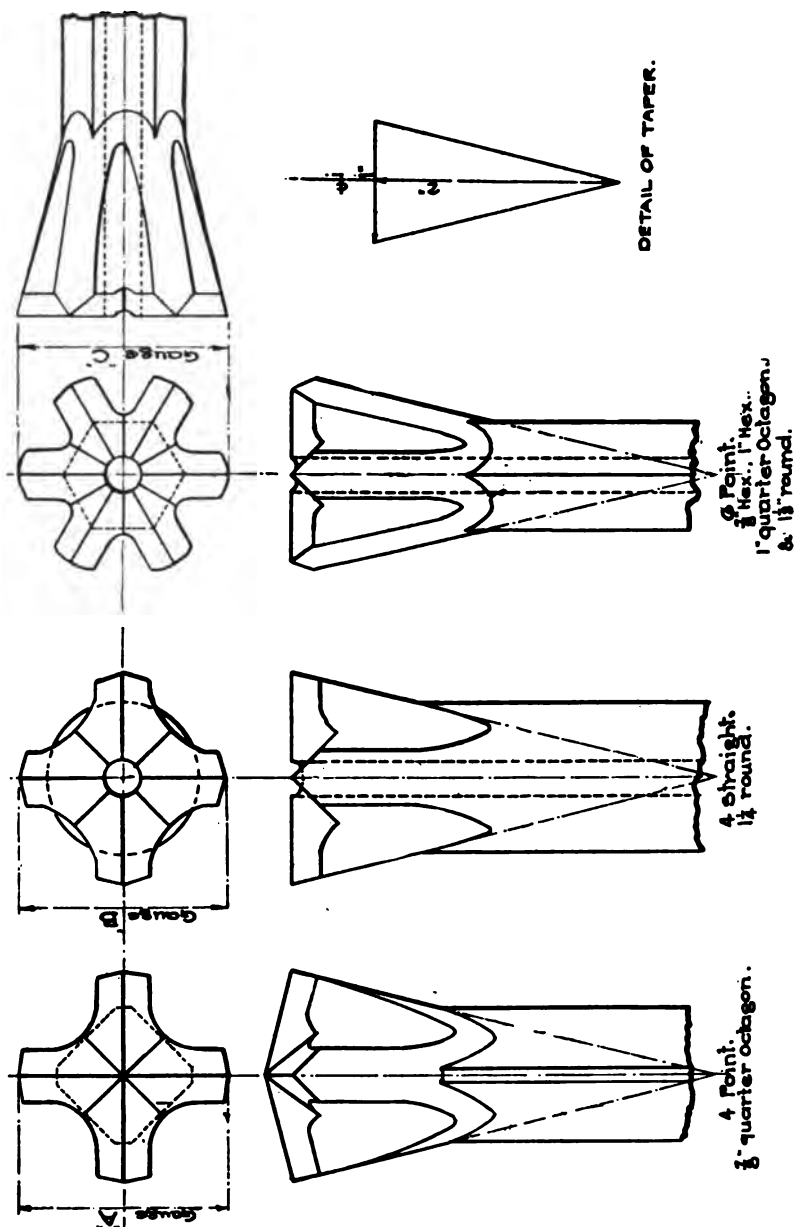


FIG. 2.—TYPES OF DRILL BITS USED AT FRANKLIN MINES OF THE NEW JERSEY ZINC CO.

degree of accuracy, and the sensitiveness of drills to air pressures and the accuracy of time and distance measurements rarely exceed this error.

It is obvious that in comparative drill tests the following factors must be considered: The nature of the rock drilled; the gauge of the drill bits and their form and condition; the maintenance of equal compressed-air pressures; similar inclination and approximate depths of drill holes; equal vigor in the rotation of hand-rotated tools; and proper fit of the drill shanks in the chuck bushings, as well as their construction so that the blows are delivered on a plane surface of proper size at right angles to the axis of the drill steel and at the center of its shank end. In the tests summarized in Table I, $1\frac{3}{4}$ in. has been taken as the standard diametral gauge of the bit, since it is a dimension which averages the gauges of drill steels used in reciprocating rock drills and is fair in determining the performances of such tools; it also represents almost the largest gauge necessary in hammer-drill stopers or block-holders, so that equal or even better performances may be expected from them as a hole is deepened with the smaller gauges in a set of drill steel. At Franklin, the testing rock is a compact coarsely crystalline white limestone, which greatly resembles a marble, and this rock proves a fair average of the various qualities of ore met in the mining operations. Although it is not hard, for a well-tempered drill bit can drill 3 or 4 ft. of hole before its cutting edges are materially dulled, and although it seems to chip freely, yet it possesses a compactness and toughness which is likely to prove surprising to one who has not previously tested a drill in it. Tests with various machines in Franklin, and elsewhere, indicate that this white limestone does not cut quite so fast as a sharp drill can achieve in Cripple Creek granite; is about on a par with Barre granite; and cuts slightly faster than Quincy granite. The chief difference is that a good drill will cut this limestone as fast in the second or third minute of its run, while it would have been dulled by the first minute's run in the granites, causing its cutting speed to fall off materially in the second and third minutes. Raised-center cross bits are the standard type used with solid steel in these tests, and flat-faced six-point bits are generally used with the hollow steels, in general the cutting speed of these bits being about the same if the rotation of the drill steel is free. Fig. 2 shows the forms of the drill bits used at the Franklin mines.

The results in Table I are not complete to date, but indicate the improvements in hammer drills, from the operators' standpoint of efficiency, during four years of advancement in the art; and it may be interesting to note that, so far as we know, no drills of the present day surpass in drilling speed and low air consumption the best drills listed in this table, although several makes of hammer drills are on a par with them. In order to avoid invidious comparisons between the different makes of drills, symbols have been used to designate each certain make and design of drill. The following abbreviations have been used in this table:

TABLE I.—Summary of Tests of Representative Drill Types Made by the New Jersey Zinc Co.

Symbol for Drill	Date	Type of Drill			Air Press., Lbs. per Sq. In. (Gauge)	Shape of Drill Bit	Free Cu. Ft. per Min.	Drill-Ing Speed, Inches per Min.	Free Air, per Inch Drilled	Factor, In. Divided by Free Air per Inch	Condition of Bit and Remarks
		Rotation	Control of Piston	Style of Feed							
A	7/27/09	Hand.	Valve.	Dir. air.	93	Raised crux.	60.4	3.56	15.62	0.246	1 corner broken, with extension.
A ₁	7/27/09	Hand.	Valve.	Dir. air.	84	Raised crux.	73.2	3.83	19.16	0.200	With extension.
B	7/27/09	Hand.	Valve.	Dir. air.	76	Raised crux.	90.0	3.20	28.20	0.118	
B ₁	7/27/09	Hand.	Valve.	Dir. air.	76	Raised crux.	76.8	4.28	17.90	0.239	Different drill of above model.
C	7/27/09	Hand.	Valve.	Dir. air.	64	Raised crux.	63.0	5.10	12.37	0.414	
D	7/27/09	Hand.	Valve.	Dir. air.	74	Raised crux.	92.7	4.50	20.60	0.218	
D ₁	7/27/09	Hand.	Valve.	Dir. air.	74	Raised crux.	86.2	5.62	15.6	0.354	Bit broken.
E	7/27/09	Hand.	Valveless.	Rev. air.	84	Raised crux.	42.0	1.93	21.7	0.089	
E ₁	7/27/09	Hand.	Valveless.	Rev. air.	68	Raised crux.	69.8	3.38	20.3	0.167	
G	10/1/09	Hand.	Valve.	Dir. air.	78	Raised crux.	75.8	5.66	13.4	0.421	
G ₁	10/8/09	Hand.	Valve.	Dir. air.	73	Raised crux.	79.5	5.91	13.4	0.450	
G ₂	10/25/09	Hand.	Valve.	Dir. air.	73	Raised crux.	79.5	6.00	13.2	0.455	
H	12/6/09	Hand.	Valve.	Dir. air.	78	Raised crux.	83.5	7.00	11.9	0.588	
H ₁	12/6/09	Hand.	Valveless.	Rev. air.	56	Flat hex.	29.8	2.46	12.1	0.203	No tappet.
I	10/21/10	Hand.	Valve.	Rev. air.	93	Crux.	66.7	2.10	31.8	0.066	
J	12/15/10	Hand.	Valve.	Rev. air.	94	Crux.	66.9	7.80	9.16	0.800	High run.
J ₁	2/25/11	Hand.	Valve.	Rev. air.	94	Crux.	68.8	9.92	9.92	0.675	Average 8 runs.
K	3/2/11	Hand.	Valve.	Rev. air.	94	Crux.	77.6	6.25	11.01	0.667	Average 23 runs.
K ₁	3/2/11	Hand.	Valve.	Rev. air.	94	Crux.	78.4	7.29	10.66	0.684	
L	4/10/11	Hand.	Valve.	Rev. air.	97	Crux.	88.0	6.90	10.90	0.683	Average 10 runs.
L ₁	4/10/11	Hand.	Valve.	Rev. air.	97	Crux.	88.0	8.29	9.21	0.683	Average 7 runs.
M	4/21/11	Hand.	Valve.	Rev. air.	93	Crux.	83.7	8.15	8.52	0.766	Average 7 runs.
M ₁	6/24/11	Hand.	Valveless.	Dir. air.	96	Raised crux.	53.5	9.35	5.67	1.600	High run to 1 1/8-in. bit.
M ₂	8/30/11	Hand.	Valveless.	Dir. air.	96	Raised crux.	58.5	11.64	5.07	2.280	Bit a little soft.
M ₃	8/30/11	Hand.	Valveless.	Dir. air.	100	Raised crux.	65.0	9.09	6.45	1.410	Average 3 other runs, same machine.
N	9/27/11	Hand.	Valveless.	Dir. air.	99	Raised crux.	62.9	10.75	6.04	1.780	
N ₁	9/27/11	Hand.	Valve.	Dir. air.	99	Raised crux.	36.0	5.28	6.82	0.774	Average 11 runs.
N ₂	10/24/11	Hand.	Valve.	Dir. air.	65	Raised crux.	66.4	4.63	8.59	0.540	Average 11 runs.
X	2/3/12	Hand.	Valve.	Dir. air.	97	Raised crux.	71.2	3.67	24.8	0.108	Drill "J," after 1 year's service.
X ₁	2/20/12	Hand.	Valveless.	Dir. air.	99	Raised crux.	63.9	6.06	11.76	0.515	High run.
X ₂	2/20/12	Hand.	Valveless.	Dir. air.	100	Raised crux.	63.5	11.10	5.66	1.963	Average 4 runs.
X ₃	3/5/12	Auto aux V.	Valveless.	Dir. air.	98	Flat hex.	79.1	10.48	6.06	1.739	Hollow bit, no air through it, high run.
X ₄	6/3/12	Auto aux V.	Valveless.	Dir. air.	87	Raised crux.	85.5	9.34	7.65	1.360	With pressure control on feed 1/4 in., bit with soft center, No. 6.
X ₅	6/3/12	Auto aux V.	Valveless.	Dir. air.	100	Raised crux.	87.0	8.50	8.95	1.068	With pressure control on feed, fine bit, No. 12.
P	2/16/12	Hand.	Valve.	Dir. air.	87	Raised crux.	67.0	8.43	10.24	0.830	Hard to rotate.
P ₁	5/13/13	Hand.	Valveless.	Dir. air.	86	Raised crux.	55.8	10.76	7.94	1.062	Rotates freely.
P ₂	5/13/13	Hand.	Valveless.	Dir. air.	86	Raised crux.	59.5	11.08	5.18	2.080	Rotates freely.
P ₃	5/13/13	Hand.	Valveless.	Dir. air.	86	Raised crux.	56.8	9.32	6.10	2.043	Rotates freely.
P ₄	5/13/13	Hand.	Valveless.	Dir. air.	86	Flat crux.	57.1	7.98	7.17	1.529	Average 9 runs.
P ₅	5/13/13	Hand.	Valveless.	Dir. air.	86	Flat crux.	57.1	7.98	7.17	1.110	Average 3 runs.
Q	5/16/13	Auto rifle.	Valve.	Rev. air.	91	Raised crux.	51.4	6.12	8.39	0.730	Hollow steel.
Q ₁	5/16/13	Auto rifle.	Valve.	Rev. air.	86	Db 16, chisel.	48.2	8.18	5.90	1.388	Air bled to control feed press.
R	9/25/13	Auto rifle.	Valve.	Rev. air.	96	Raised crux.	112.0	6.85	16.25	0.421	Air bled to control feed press.
R ₁	9/25/13	Auto rifle.	Valve.	Rev. air.	86	Flat hex.	102.2	7.80	13.63	0.560	Air bled to control feed press.

General.—Air measured by displacement of water. Rock was white crystalline limestone. Gauge of bits was 1 1/8 in. to 1 1/4 in.

Auto aux V = automatic auxiliary valveless control of rotations.

Auto rifle = automatic rotation caused by a piston reciprocating as though it were controlled by a rifle bar.

Dir. air = direct-air feed, or one in which the feed cylinder is rigidly attached to the hammer cylinder and in which the feed piston or plunger extends from the rear end of the machine by virtue of the air pressure applied to it.

Rev. air = reversed-air feed, or one in which the feed piston is rigidly attached to the hammer cylinder and the feed cylinder is free to extend backward, so readily adapts itself to the customary column mounting of stopping drills.

Some tests were included in this table for the consideration of points to be made later. Before studying the improvements in hammer-drill efficiency it seems wise to explain the reasons for offering the figures in the last column of figures as representing a factor of "drill desirability."

In determining the relative merits of rock drills, whether of the reciprocating or hammer type, the logical basis is one of cost. Therefore, the drill which bores a foot of drill hole of standard cross-section at the lowest cost rate for drilling labor, power, and maintenance (including amortization), would have the highest "factor of desirability;" and a formula to express this may be developed as follows:

Let

F be the "factor of desirability,"

D be the cost of drilling labor per foot of hole,

P be the cost of power per foot of hole,

M be the cost of maintenance per foot of hole.

Then,

$$F = \frac{1}{D + P + M}$$

Let

t = Period of time for drilling-speed test, in minutes.

d = Depth of hole drilled in time, t , in inches.

$S = \frac{d}{t}$ = Drilling speed during actual running of machine, in inches per minute.

L = Hourly wage of drilling labor, in cents.

O = Percentage of time spent in drilling to total operating time including the changing of drill steels and shifting to new positions and starting of new holes.

Then

$$D = \frac{L}{60SO} = \frac{L}{5dO} = 0.2 \frac{tL}{dO}$$

Let

p = Power cost to produce 100 cu. ft. of free air, compressed to standard drill-testing pressure, in cents.

v = Number of cubic feet of free air used in test by operating drill.

d = Depth of standard hole drilled, in inches.

Then

$$P = \frac{12pv}{100d} = 0.12 \frac{pv}{d}$$

Substituting these values of D and P in the original equation we obtain

$$F = \frac{1}{0.20 \frac{tL}{dO} + 0.12 \frac{pv}{d} + M} = \frac{d}{\frac{0.2L}{O}t + 0.12pv + dM}$$

Since L is a constant for any particular mine, and O for a given number of steel changes with any particular type of drill—such as a column-mounted reciprocating drill, a column-mounted hammer drill, an air-feed hammer drill, a block-holing drill, etc.—we may simplify the equation by substituting

$$k = \text{coefficient of drilling} = \frac{0.20L}{O}$$

Also, since p is a constant for any particular mine we may further simplify by placing

$$k' = \text{coefficient of power} = 0.12p$$

and we then have the general equation for any particular mining conditions and type of drill

$$F = \frac{d}{kt + k'v + dM}$$

However, the correct value for maintenance and amortization of any particular type and make of drill can be determined only after operations extending over months or years, so that this factor may well be left out of a formula which is to be used for classifying drills after drilling speed and power consumption tests, which may be completed in a short time. The consideration of the reduction of drilling speed and the increase of power consumption, which occur in a drill because of wear or any other normal results of service, may fairly be placed in the same class as maintenance. Judgment as to the materials, workmanship, and design of any drill, as well as reports of its satisfactory service elsewhere, will lead to a rough estimate of the final desirability of a drill if it has shown a high standard, on testing based on drilling speed and power consumption.

The equation is thus simplified to the form

$$F = \frac{d}{kt + k'v}$$

But other highly important factors enter into the problem of selection of a drill, namely, the reduction in labor units, capital, and overhead charges brought about by an increased drilling speed and increased tonnage per machine; the increased efficiency of supervision and work caused by the reduction and concentration of the number of working places; the possibility of producing a greater tonnage from any property with a limited number of working places; and the possibility of reducing the drilling equipment, with its attendant stock of spares, hoses, and connections, and extensive air mains, if a drill with a greater drilling speed may be employed. It therefore seems that the following formula is more indicative of the actual merits of drills, although theoretically it has no derivation, and must be considered empirical; it also possesses the virtue of reducing to a simple form. This formula for a "factor of desirability" has been used for the past six years at the Franklin Furnace mines of the New Jersey Zinc Co. All coefficients have been omitted since the following drill tests have all been under the same standard conditions.

$$F' = \frac{1}{DPM}$$

Since M is treated separately, as has been previously suggested, the equation becomes

$$F' = \frac{1}{DP}$$

Now if the same values previously deduced for D and P are substituted,

$$F' = \frac{1}{\frac{kt}{d} \times \frac{k'v}{d}} = K \frac{d^2}{tv}$$

where K is a new coefficient equal to the reciprocal of the product of k and k' .

Therefore, the "factor of desirability" equals the drilling speed, in inches per minute, divided by the power consumption, in cubic feet of free air, per inch drilled. It is quite evident that the factor gained from the quotient of inches drilled per minute divided by cubic feet of free air per minute (or the reciprocal of this quotient) gives merely the power consumption per inch of hole drilled and ignores the quantity of drilling which may be accomplished.

The application of both of these formulas for F and F' to a hypothetical problem may be of interest to show the comparative results within the limits of practice.

Let us assume that 30 h.p. is required to compress 100 cu. ft. of free air per minute to 100 lb. per square inch gauge pressure and deliver the same to a drill in the mine; that the power cost is 1c. per horsepower-hour;

that a drill which shows a drilling speed of 10 in. per minute on test verages 20 ft. per hour under working conditions, and uses 60 cu. ft. of free air per minute on test; that another drill will show a drilling speed of 6 in. per minute on test with an air consumption of 36 cu. ft. per minute and will average 12 ft. per hour under working conditions, and that the wage scale for drill runners is 40c. per hour, then,

For the fast drill:

$$t = 1 \text{ min.}$$

$$d_1 = 10 \text{ in.}$$

$$L = 40\text{c.}$$

$$O_1 = 20 \div \frac{10 \times 60}{12} = 0.40$$

$$p = \frac{30 \times 1}{60} = 0.5\text{c.}$$

$$v_1 = 60$$

$$F_1 = \frac{d}{0.2 \frac{L}{O_1} t + 0.12 p v_1} = \frac{10}{\frac{0.2 \times 40 \times 1}{0.40} + 0.12 \times 0.5 \times 60} = 0.424$$

$$F'_1 = \frac{1}{\frac{0.20 t L}{O_1 d_1} \times \frac{0.12 p v_1}{d_1}} = \frac{O_1 d_1^2}{0.024 t L p v_1}$$

$$= \frac{0.40 \times 100}{0.024 \times 1 \times 40 \times 0.5 \times 60} = 1.389$$

For the slow drill:

$$d_2 = 6 \text{ in.}$$

$$v_2 = 36 \text{ cu. ft.}$$

$$O_2 = 12 \div \frac{6 \times 60}{12} = 0.40$$

$$F_2 = \frac{6}{\frac{0.2 \times 40 \times 1}{0.40} + 0.12 \times 0.5 \times 36} = 0.271$$

$$F'_2 = \frac{0.40 \times 36}{0.024 \times 1 \times 40 \times 0.5 \times 36} = 0.833$$

Thus the relative factors for the two drills by the first formula have a ratio of 0.424 to 0.271 or 1.56 to 1; and by the second formula (empirical) the ratio of factors is 1.389 to 0.833 or 1.67 to 1. In other words, by the empirical formula the fast drill is credited with about a 7 per cent. higher rating than by the theoretical formula, and this does not seem an undue allowance to cover the unestimated advantages previously enumerated.

Records made previous to July, 1909, have not been shown in Table I since much of the work done in 1907 and 1908 was distinctly experimental in determining the desirable cylinder diameters, lengths of strokes,

piston weights, valve weights, etc., but such records show drilling speeds of about 2 to 3 in. per minute with air consumptions of from 40 to 70 cu. ft. of free air per minute at 90 lb. per square inch gauge pressure. The listed tests made during 1909 cover most of the well-known American makes of hammer drills at that time, and if one excepts the drills denoted by symbols G , G_1 , etc., since they were experimental tools, the design of which was developed by the New Jersey Zinc Co. at Franklin Furnace, N. J., it is noticeable that about $4\frac{1}{2}$ and 5 in. were the highest drilling speeds obtainable at about 90 lb. pressure and with an air consumption of 60 to 90 cu. ft. of free air per minute; and for various drills the "factor" varied from 0.09 to 0.41. Those drills marked G , which were made exclusively for the New Jersey Zinc Co., increased the drilling speed about 40 per cent. above the best previous drill performances, and remained unequaled in drilling speed for a year and unsurpassed for about a year and a half. The fact that a number of these drills were included in the equipment at Franklin accounts for part of the increased stoping efficiency during the year 1911, as cited later. Although it was then the opinion of some unprejudiced persons, well versed in the drilling art, that such tools had reached their practical limit of drilling speed as well as the limit of strengths of materials, yet 18 months later a new type of drill was developed to achieve 20 per cent. more drilling with twice as good a factor, and a renewed equipment of these other drills again increased the mining efficiency. Again a period of 18 months sufficed for the production of a hammer drill which still further advanced the drilling speeds 20 per cent., and since the introduction of this drill we have been able to find several drills which surpassed it 10 to 20 per cent. in drilling speed.

In Table I some seemingly freak runs are noticeable, which are included to call attention to the variability of results in presumably standard testing. For instance, under drills D it appears that a bit with two wings broken will drill faster and at a lower air consumption per minute than can be attained with a perfect bit; and again, with drill M_1 , a bit which has proved a little soft and battered drills one-fourth more per minute than bits in proper condition and with the same air consumption. Furthermore, the tests of one person indicate that, when the size and form of the drill bits are the same, faster drilling can be done with short steels than long ones, while another investigator shows a greater drilling speed with long steel than with short. The use of tappets or anvil blocks between the shanks of drill steels and pistons is generally estimated as causing a reduction of 20 to 30 per cent. in the drilling ability, but some tests do not confirm this and show even an increased cutting speed with the use of anvil blocks in a machine otherwise the same. With some drills the use of water to clean the cuttings from the hole seems to cause a cutting speed below that obtainable through the use of

compressed air for the same purpose, but in other instances the advantages are reversed. In short, there seem to be so many variables in the drilling problem as to warrant a 10 per cent. variation in the results of supposedly standard tests, and a number of runs should be made to gain a fair average; or strict judgment of machines should not be made within this limit.

Perhaps the consideration of the physical phenomena relating to the process of drilling may prove of interest and value. When rock is excavated by a drill bit three applications of forces seem to be involved—by abrasion, by crushing, and by severing or chipping. Although all of these must take place to a certain degree, the greatest amount of useful work is performed when the percentage of force applied to chip reaches a maximum. But in rock it appears that chips can be produced in radically different ways: first, by the severing of molecules, and second, by the reflex forces produced in an elastic medium. To illustrate this, consider the chipping of a comparatively inelastic substance such as lead. With a hammer and a chisel, whose axis is inclined considerably from the normal to the surface of a lead block, it is possible to sever the lead and roll up chips, but if the chisel is normal to the surface of a thick block only an indentation can be made and there probably will be a raised area about the indentation to accommodate a certain percentage of the displaced metal. On the other hand, with a highly elastic material, such as glass, the forces impressed by a normally positioned chisel will cause a compression of the molecules, whose elasticity will cause their expansion toward a free, unresisted surface. Since the greatest forces are developed at the surface, since the penetration of the chisel carries some forces to a depth below the surface; and since the chisel surface itself applies some forces at an angle to its axis and impedes the re-expansion of molecules to the space it occupies, therefore, the reflex forces produce more or less cone-shaped chips or flakes and leave a corresponding crater in the block of glass. Now, if the chisel is placed near the edge of a block of glass, the blow upon it will induce stresses to another free face and a correspondingly larger chip will be produced because of the tendency of the forces to seek relief in the shortest direction as well as because of the severing effect. The method of cutting of a drill bit is commonly shown as taking place in this last way with the progressive chipping of a series of benches or steps, but it is doubtful whether such a procedure exists, except in rare instances, for the speed and latitude of rotation between consecutive blows of the drill piston or hammer cannot be controlled with sufficient precision nor adjusted to the various rocks; and an inspection of the cuttings from a drill hole shows them to be flakes, or a crushed and abraded powder.

In the formation of these flaky chips there may be a limiting force of blow for each velocity of impression in order to gain the most useful

work (*i.e.*, in the production of flakes), for it appears that beyond certain limits the blows increase the percentage of crushed material and the drilling speed does not vary with the force applied, so that some heavy-hitting drills accomplish more in medium-soft ground when a portion of their blows are absorbed by a tappet at the shank end of the steel or by a cushion of water intervening between the bit and the rock. If the force of the blows was lessened by a reduction in air pressure the speed of the piston would be slowed up, and the drilling would suffer from the fewer number of blows per minute.

The transmission of the kinetic energy of the piston to the rock is also influenced by many factors. The blow may be delivered against the rock by the free drill steel which is driven forward through the intervening air or water by the impact of the piston and the velocity of the steel will depend upon the relative masses of the drill steel and piston, the velocity of the piston, and the coefficient of elasticity of the steel, in accordance with the well-known laws of mechanics which deal with elastic or partly elastic bodies and their impact. The drill steel in this way assumes the functions of a "jumper" drill which is driven against and rebounds from the rock at a high frequency, and its action is well seen in most all screw-feed hammer drills with the ringing or jingling of the steel in a drill hole. Another mode of force transmission is by compressional waves, traveling through the drill steel from the shank to the bit. This latter condition brings a cutting effect only when one end of the steel is tight against the rock, but then proves very efficient. Although the air-feed hammer drills usually chatter the steel against the rock, like a projectile shot from the chuck bushing by impacts of the piston, yet it seems possible to approximate the other working condition by designing the air feed so that the pressure is lowered as the piston is traveling on its back stroke (possibly by taking the supply air, for the back stroke, from the air feed) and so that the air-feed pressure builds up and forces the drill against the rock just before it is struck by the piston. The reversed-air feeds may sometimes approximate these conditions and then assist the machine to a higher drilling speed. If hammer drills were made so that the drill steels were always held firmly against the rock, when the piston strikes them, it seems unquestionable that the greatest efficiency of the blows of the piston would result; providing they were properly timed, for no energy would be lost by reason of the inertia of the drill steel, but only that due to heating, resulting from the imperfect elasticity of the metal. The question of the proper timing of the piston blows opens another phase of the matter, namely, the reaction of the rock upon the drill steel; and this effect is the more pronounced with harder rock. It tends to speed up the piston and is so noticeable in running a machine against a metal block as to invalidate, as too high, all air-consumption tests so conducted. The effect of these reactive vibrations upon the drill steel may prove very

marked and serious. Where the reactive vibrations interfere with oncoming compressional waves, considerable energy is dissipated, and at times one may be so fortunate as to detect points of increased temperature (probable nodes) upon a drill steel which is cutting ground; and it is no uncommon thing to see a drill steel, in service, break at two points (into three pieces) simultaneously, probably from fatigue because of these vibratory stresses. On the other hand, if these vibrations synchronize at the bit it is quite possible that the chipping forces are greatly augmented, and such an explanation may readily answer those puzzling drill tests in which a dull or broken bit exceeds a finely formed bit in drilling speed. For a long time at Franklin a tally was kept of the different individual drill steels which entered into the testing, with the hope of determining that some particular piece of steel produced the greatest cutting speed, but no conclusion could be drawn from the records, except that the changes in length due to resharpening probably masked any possibility of determining the suitable lengths for maximum efficiency. And it seems quite plausible that such a result should be expected if the possible wave lengths of the compressional vibrations in the drill steels are considered. Probably these reactive vibrations occur to a great extent, as well, in the process of drilling, where the steel dances in the chuck and against the rock, for steel breakage appears equally as high, if not higher, with such a type of machine as with the pneumatic feed, and tests comparing these two types for such effects might prove very interesting as well as instructive.

But still other factors influence the force delivered at the rock. If the anvil block or tappet is not in contact with the drill when the piston strikes, a considerable energy loss occurs through the transference of momentum to several pieces. If the steel is bound in the chuck bushing, a great amount of the energy is absorbed by the friction. If the steel is not straight, it loses energy because of the flexure. If the chuck is badly worn, the axis of the steel does not coincide with that of the drill and there is a loss due to the oblique, eccentric impact. If the steel is tight in the drill hole, or if the friction against the side of the hole is great because of its depth, the velocity of the steel, as a projected body, is lessened and the drilling speed is reduced.

The length of the drill steel is an item generally credited as an important influence, and common opinion supports the idea that the cutting speed falls as the length of the steel increases, although some people, on the contrary, feel sure that the long steels drill the fastest. The tests conducted at Franklin do not lend an unqualified support to either view, for the peculiarities of different types of machines play so important a part. For example, if the air feed is very strong in a stoping drill the additional counteracting weight of a long and heavy steel may so improve the working conditions as to indicate a superiority for the long steel, and if

the air feed is weak the reverse may be true; if the drill steel cuts by virtue of a dancing or "jumper" action, the mass added with length may so reduce its velocity against the rock as to bring it below the amount required for efficient chipping; if the piston normally delivers too heavy a blow for the rock, the drilling speed may be improved by the added inertia of the long steel; and if the steel is always against the rock when a blow is delivered, it is doubtful whether the length of the steel plays an important part unless the permitted decrease in the gauge of the drill bits aids the cutting speeds. It is, of course, to be understood that the above considerations of drill-steel lengths refer to the performances with bit gauges of the same diameter.

The use of an anvil block is considered by some drill designers to necessitate a loss of from 20 to 30 per cent. of the power of a drill, but actual tests do not always indicate such a condition when the identical steel is tested in the same drill with and without a tappet. The results probably depend upon how frequently the tappet is struck when away from the shank of the steel, and also upon the suitability of the machine to the rock, for if its blows are too heavy the intervention of a loose tappet might reduce their force, with a benefit in drilling speed. The use of water at the bottom of the hole ordinarily consumes about 10 per cent. of the cutting speed if there is no tendency for the drill bits to lose their temper, and compressed air for cleaning the holes encourages a greater drilling speed, providing the cushion of water in the bottom of the hole does not have a benign influence in reducing too powerful a blow upon the rock.

The manner in which a drill is rotated has a bearing upon the amount of work accomplished, and with hand-rotated tools, a vigorous rotation with a rapid and wide arc of swing produces the best results; with power-rotated drills it is possible to reach such a speed as to abrade and dull the drill bits, and consequently lessen the drilling speed. It seemed that, with a positive and constant rotation, the axial planes of the cutting edges of the drill bits should be at the same angle with the cut surface as the resultant velocity vector, as estimated for the rotative and striking velocities; and such a bit was tried at Franklin without showing a change in cutting speed, probably because with either bit the chips came out in flakes, as previously described.

In view of the fact that the subject of hammer drills is more or less in its infancy and literature in regard to them is rather limited, it seems desirable to correct at the earliest opportunity any typographical or other errors which, if accepted without investigation, might work to the detriment of the art of drilling. In this connection it seems that some statements should be corrected in the 1910 edition of Eustace M. Weston's book, *Rock Drills*, in the chapter Philosophy of Process of

Drilling Rock, under the sub-heading of hammer drills. In considering the kinetic energy of a blow he states, on p. 139:

"In other words, to double the energy of a blow it would be necessary to double the mass, or weight, if the velocity is the same; but to double the energy, keeping the mass the same, the velocity must be increased four times. The weight of the piston hammer of the largest type of drill is 15 lb. The weight of piston, steel, etc., of a piston drill varies from 60 to 125 lb., so that a blow of equal force can be delivered by a hammer drill only by increasing the velocity of the hammer very greatly. This is acknowledged, for as one hammer-drill maker states, the weight of the piston is one-fourth that of a piston drill; but the velocity is four times as great. To give a blow equal in power it should be sixteen times as great."

A mathematical error appears to have been made in the premises of Mr. Weston's argument and his consequent deductions as to the practical impossibility of hammer drills being able to compete with piston drills are quite logical, but probably at fault.

If the kinetic energy of a body, such as a drill piston, is designated by K , its velocity by V , and its mass by M , then,

$$K = \frac{1}{2}MV^2 \text{ and } K_1 = \frac{1}{2}M_1V_1^2$$

Now if M_1 equals M and K_1 is, say, twice the value K , then,

$$V_1^2 = 2V^2 \text{ and } V_1 = V\sqrt{2}$$

therefore,

$$V_1 = 1.414V$$

So the velocity of the piston in a hammer drill need be only 1.4 times as great as that when the kinetic energy of the piston is cut in half.

Again, in the example comparing the piston weights of piston and hammer drills, Mr. Weston appears in error in stating that the velocity should be 16 times as great, for if the piston of a hammer drill is one-fourth the weight of that of the piston drill the velocity of the hammer-drill piston need be only twice as great as that of the piston drill in order to deliver blows of the same energy; and the hammer drill will also surpass the piston drill since it will strike twice as many of such blows per minute. The necessity of using high air pressures in hammer drills is only incident to the peculiarity of certain drill designs and is not dependent upon the divorcing of the piston from the steel. If we are to consider the shock upon the parts of two drills of equal capacity it is evident that with the shorter piston strokes in hammer drills, with the increased number of blows, whose final striking velocity is equal to that of a piston drill under comparison, the weight of the hammer-drill piston may be less and the energy in each individual blow may be less in order that the same amount of energy per minute be developed. Therefore, the shocks upon hammer-drill parts are more frequent but not as heavy as the shocks upon piston drills of equal capacity.

It is extremely difficult to get adequate figures as to the maintenance of drills unless some special forms are kept, which become to all intents a ledger account of each individual drill, for questions naturally arise as to the cost per foot of hole drilled, the length of time the machine has been in service and has been running, the number of holes drilled, the kind of rock encountered, and the supply of steel used, as well as the drill parts replaced. The New Jersey Zinc Co. uses a system of punched slips for shift bosses' reports and "drill record" slips are a part of the scheme. These are punched in duplicate by the shift boss and filed at the mine office and main office, where the information is transferred to large sheets, of which each one accommodates the record of one machine for a month, and the footings are carried forward so as to indicate the total work accomplished and maintenance of any machine "to date." The repair parts are designated as to whether they are new or old (second-hand) ones, and

DRILL RECORD																				AB 1507-500 B's 11-14									
		STOPE		LEV.																									
JAN.	SEP.	0	DR'L'G	HOLES	0	0	1/2	0	*	R	H	1st	2nd	3rd	4th	5th	6th	TYPE:-	NO.										
FEB.	OCT.	1	HRS	MIN	0	0	1/2	0	*	R	H	1st	2nd	3rd	4th	5th	6th	TYPE:-	DISCH'D:-										
MAR.	NOV.	2	1	10	1	1	1	1	1	1	1	1	1	1	1	1	1	REC'D:-	DISCH'D:-										
APR.	DEC.	3	2	20	2	2	2	2	2	2	2	2	2	2	2	2	2	SENT MACH. SHOP:-	F.W.										
MAY		4	3	30	3	3	3	3	3	3	3	3	3	3	3	3	3	REP. PARTS:-	N O										
JUNE		1	5	4	4	4	4	4	4	4	4	4	4	4	4	4	4		N O										
JULY		2	6	50	5	5	5	5	5	5	5	5	5	5	5	5	5		N O										
AUG.		3	7	6	6	6	6	6	6	6	6	6	6	6	6	6	6	REMARKS:-	N										
1914		8	7	P	7	7	7	7	7	7	7	7	7	7	7	7	7		S										
1915	DAY	9	8	ND	8	8	8	8	8	8	8	8	8	8	8	8	8		E										
1916	WRT	10	9	ND	9	9	9	9	9	9	9	9	9	9	9	9	9		W										

FIG. 3.—DRILL-RECORD SLIP.

original and subsequent drilling-test records are noted on the same summary sheet.

Figs. 3 and 4 show the record slip and sheet. The locations of drills are by stope-slice co-ordinates, the class of work (whether raising, drifting, stoping, block-holing, or drilling chutes) is indicated, the kind of rock (ore, limestone, gneiss, pegmatite, garnet, or feldspar) is punched, and if the machine is idle, broken, or being cleaned those points are recorded.

The compressed-air rock drill made revolutionary changes in mining methods and in the reduction of mining costs in units of labor per ton of ore, and at Franklin even more marked savings have been made through the development of hammer drills. There, in the days of hand drilling, a total of 8 ft. in three drill holes with varying diameters of $1\frac{3}{4}$ to $1\frac{1}{4}$ in. was considered a fair 10-hr. shift's work, and possibly 8 tons of ore would be broken per drill-shift or 4 tons per man-shift. With 3-in. reciprocating rock drills from 20 to 40 ft. of drill holes, ranging in diameter

from $2\frac{1}{2}$ to $1\frac{1}{2}$ in., would be the average work for a 10-hr. shift, although on rare occasions some men might drill as much as 80 or 90 ft. of holes in a shift, and possibly 20 tons of ore would be broken per shift, or 10 tons per man-shift, since two men were needed on a drill. It seems

THE NEW JERSEY ZINC COMPANY
FRANKLIN, N. J.

A B May 1906

DRILL RECORD

MAKE: _____ TYPE: _____ SHOP No. _____

PURCHASED ON REQ.; A B _____ IN SERVICE: _____ IN _____

CYLINDER DIAM. _____ inches PISTON STROKE _____ inches PISTON WEIGHT _____ lbs. oz.

AVG. DRILLING SPEED (to date) _____ inches per min. AVERAGE AIR CONSUMPTION _____ cu. ft. per min. free FACTOR _____

ORIGINAL _____ DRILLING SPEED _____ inches per min. AIR CONSUMPTION _____ cu. ft. per min. free FACTOR _____

PRESENT _____ DRILLING SPEED _____ inches per min. AIR CONSUMPTION _____ cu. ft. per min. free FACTOR _____

DATE	WORKING PLACE		DRILLING TIME		Holes Drilled	Footage Drilled	Drilling Rate (ft. per hr.)	Material Drilled	PCL. DRILL STEEL USED					HRS. REPAIR LABOR		REPAIR PARTS (Make's Symbols)	M'T'OE CHARGES		
	Loc.	Loc.	Hrs.	Min.					1st	2nd	3rd	4th	5th	Special	\$2.25		\$1.25	SUPPLIES	REPAIR LABOR
Brought Over																			
1st DAY																			
2nd DAY																			
3rd DAY																			
4th DAY																			
5th DAY																			
6th DAY																			
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29th DAY																			
30th DAY																			
31st DAY																			
TOTAL FOR MONTH																			
Carried Forward																			

FIG. 4.—SHEET FOR RECORDING DRILL PERFORMANCE AND COSTS.

that as a rule a greater tonnage per foot of hole was obtained with hand drilling because of the fact that, rather than dismount and reset heavy drill columns, machine men would tend to place as many holes as possible from one set-up, therefore many holes were placed disadvantageously

for breaking efficiency. Another cause, which would contribute to the same results, would be the difficulty of starting holes with piston drills on uneven sloping faces, so that holes were frequently deflected from the direction in which they were supposed to be placed. These figures would lead to the rough estimate that $2\frac{1}{2}$ times as much tonnage per drilling man-shift was accomplished by piston drills as by hand drilling.

With hammer drills 80 to 100 ft. of $1\frac{3}{4}$ to $1\frac{1}{4}$ in. drill holes are placed by one man in a 10-hr. shift, about 150 to 200 tons of ore will be broken per drill-shift and the same amount per drilling man-shift, or 15 to 20 times the amount broken per man with reciprocating drills. Of course the entire credit for such increase in tonnage cannot be given to the type of drill, for improved organization, system of working, and supervision have undoubtedly played an important part; but the greater mobility and flexibility of the light hammer drills have permitted and encouraged a more efficient placing of drill holes; have cut in half the labor necessary to run a drill; and permitted a more effective supervision and mining scheme. The actual tonnage broken per man working in a stope will not be so high, comparatively, since it has been found worth while to place additional men in stopes to sledge and block-hole large chunks of ore, which were formerly often allowed to become buried and so proved obstacles to high tramming efficiency by blocking chutes through which the shrinkage stopes were drawn into tram cars.

Table II shows the gains which have been made with the adoption of hammer drills by the New Jersey Zinc Co. It is to be regretted that no records of tonnage and labor were available for earlier years, so as to cover the average efficiencies before the advent of hammer drills and back to the days of hand drilling. The different divisions of mining work are classified in this table as drifting, raising, stoping, and open cut or quarry, and it may be interesting to summarize the important features, reducing the labor to an hourly basis, inasmuch as a change was made from a 10-hr. to an 8-hr. shift basis in July, 1913.

Drifting.—There have been no radical changes in the placing of the drill holes in drifts since the adoption of the air-feed hammer drill for this work, but one man with a single machine is now placed in a heading; he is instructed to "pull" a "round" each 8-hr. shift, stopping overtime if necessary, and to accomplish an advance of $3\frac{1}{2}$ to 4 ft. per round. Two men operating a reciprocating rock drill formerly made an advance of a 5 to 6 ft. round in five 10-hr. shifts. So the drilling labor (runners and helpers) per foot of advance averages 18.4 hr. for the entire mine during the year 1910, when reciprocating rock drills were solely in use. As shown by the average for 1913, hammer drills have reduced this figure to 5.7 hr. per foot of advance, or about one-third the former labor of drilling and blasting. The explosive costs have also been reduced by

the use of hammer drills from the figure of \$1.84 per foot of drift during 1910 to \$1.40 per foot in 1913, for two probable reasons.

First, hammer drills permit the placing of drill holes smaller in diameter than those bored by reciprocating drills, so that an unnecessary amount of explosive is not required merely to fill the holes sufficiently to distribute the force of the explosion.

Second, the flexibility and ease of rigging the light hammer drills permit and encourage a more efficient placing of drill holes. The almost exclusive use of 1 by 8 in. explosive cartridges now, as contrasted with the $1\frac{1}{8}$ by 8 in. cartridges formerly used, demonstrates the first contention, for in terms of 1-in. powder, the equivalent of 36.8 sticks per foot of drift was used in 1910, and 34.6 sticks per foot in 1913. The drill shifts per foot of advance have been lowered from 0.44 in 1910 to 0.20 in 1913, and the corresponding drill hours from 4.4. to 1.8.

The different drifts may vary in size from 6 by 7 ft. to 8 by 11 ft. in section, and perhaps 7 by 8 ft. is an average section. Because of the compact, tough nature of the ground, it requires from 20 to 30 drill holes in a round, and 24 would be a fair average, so the drilling operation is an important factor of the drifting costs. The following comparison of the average drifting costs for each year shows the saving which has been possible because of hammer drills; but only the cost of drilling labor and explosives is considered. The record drift in 1913 was driven for \$2.06 per foot.

	1910	1911	1912	1913
Drifting cost per foot.....	\$5.33	\$4.92	\$3.35	\$2.70

Raising.—In 1908, using $2\frac{1}{4}$ -in. piston reciprocating rock drills, 0.7 ft. of 6 by 6 ft. raise per 10-hr. drill shift was made with a labor expense of 28.5 man-hours per foot of raise. About 27 ft. of drill holes were placed per shift, 24 holes were placed in a round, and 16 lb. of explosives were used per foot of raise advance, at a cost of \$2.70 per foot for supplies. Since labor was then paid \$2 and \$1.55 per 10-hr. shift, the total cost of raising was approximately \$7.50 per foot of advance.

During the same year, 1908, hammer drills were introduced, and an advance of about 1.5 ft. per drill-shift was made with a labor expense of 13.3 man-hours per foot of advance. About 50 ft. of drill holes were placed per shift, 24 holes per 5-ft. round, and 10 lb. of explosives were used per foot of advance, at a cost of \$1.75 per foot for supplies and a total cost of \$4.10 per foot of raise, or only 55 per cent. of the cost with the reciprocating rock drills.

The development of hammer drills with increased drilling speed permitted the reduction of the drilling labor to 7.8 hr. per foot of raise advance, and the explosives cost to \$1.54 per foot of raising done in the year 1910; and a further reduction to 4.8 hr. of drilling labor during 1912,

TABLE II.—Annual Comparisons of Mining Efficiencies with Piston Drills and Hammer Drills

Date	Drifting										
Year	Footage Advance	Drill Shifts per Foot Advance	Type of Drills, Per Cent.		Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Foot	Powder in Sticks per		Explos. Cost per Ft. Advance	Runners and Helpers per Foot Advance
			H. D.	P.	1½ in.	1 in.		Foot Advance	Drill Shifts		
1908 ^a	494	1.45						36.4	25.2		
1909	4,890	1.31						32.7	25.0		
1910	4,909	0.74		99.2	73	27	5.43	31.1	42.5	\$1.84	1.84
1911	2,814	0.63		99.2	96	4	4.95	26.5	41.9	\$1.59	1.71
1912	374	0.44	100		11	89	7.40	36.4	89.6	\$1.61	0.84
1913	905	0.33	100			100	5.56	34.6	106.0	\$1.40	0.59

^a Last 4 months.

Date	Raising										
Year	Footage Advance	Drill Shifts per Foot Advance	Type of Drills, Per Cent.		Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Foot	Powder in Sticks per		Explos. Cost per Ft. Advance	Runners and Helpers per Foot Advance
			H. D.	P.	1½ in.	1 in.		Foot Advance	Drill Shifts		
1908	715	1.18						27.7	23.4		
1909	5,446	0.90						29.3	32.4		
1910	4,257	0.44	97.0	...	16	84	5.11	27.4	62.7	\$1.54	0.78
1911	1,865	0.23	98.5	...	8	92	4.38	21.5	92.5	\$1.04	0.65
1912	1,311	0.17	100.0	...	2	98	3.46	20.1	115.0	\$0.93	0.48
1913	2,306	0.20	100.0	...	1	99	3.94	21.4	106.2	\$1.03	0.58

Date	Stopping (active)												
Year	Net Tons Ore Broken	Type of Drills, Per Cent.			Net Tons Broken, Excl. B. H. Drills per Drill Shift	Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Foot	Powder in Sticks per		Explos. Cost per Net Ton Broken	Net Tons Broken per Man in Stope	Remarks
		B. H.	H. D.	P.		1½ in.	1 in.		Net Ton	Drill Excl. B. H. Shift			
1908	57,000 ^b				23.4				0.76	17.7			
										19.3			
1909	361,000	7.0	24.5	68.5	20.5				1.01	20.7			
1910	337,000	12.7	62.3	25.0	38.1	16	84	0.310	1.025	39.0	\$0.055	13.2	
1911	386,000	30.1	55.8	14.1	121.0	11	89	0.356	0.77	95.8	\$0.041	20.3	
1912	427,000	35.5	62.0	2.5	195.0	6	94	0.415	0.86	108.3	\$0.045	25.6	
1913	494,330	30.3	69.7		170.0	2	98	0.420	0.89	233.0	\$0.049	26.7	

^b Two months, estimated.

Date	Filling													
Year	Net Tons Ore Broken	Type of Drills, Per Cent. $\frac{1}{2}$ in.			Net Tons Broken, Excl. B. H. Drills per Drill Shift	Per Cent. Size of Powder, 50 Per Cent. Strength		No. Caps per Foot	Powder in Sticks per		Explosive Cost per Net Ton Broken	Net Tons Broken per Man in Stope	Remarks	
		B. H.	H. D.	P.		1 1/4 in.	1 in.		Net Ton	Drill Excl. B. H. Shift				
1910	25,400	58.6	14.2	27.2	114.2	28	72	0.922	0.745	(35.2) ^c 85.3	\$0.051	24.4 ^c 4.9		
1911	64,500	79.2	1.6	19.2	355.0	29	71	0.787	0.614	(45.3) 278.0	\$0.042	62.0 50.6		
1912	107,800	62.0	11.4	26.6	228.3	29	71	0.571	0.719	(62.4) 164.5	\$0.040	54.2 46.0		
1913	250,490	45.7	54.3		236.0	39	61	0.524	0.712	(91.3) 168.0	\$0.042	82.2 51.0		

^c Bracket figures include block-holing shifts; others rated against runners only.

and an explosive cost of \$0.93 per foot, although the wages were \$2.20 and \$1.70 per 10-hr. shift. These costs rose slightly in 1913, since wages rose to \$2.25 and \$1.85 for 10-hr. shifts, and in July of the same year the working hours were lessened from 10 to 8 and the hourly wage was increased to \$0.281 and \$0.231. However, the cost per foot was then only 5.2 hr. of drilling labor and \$1.03 per foot for explosives. About 18 drill holes are now placed to pull a 5-ft. round and two men are expected to blast a round each 8-hr. shift and are each paid 11 hours' time for performing the task.

The average raising costs for operating labor and explosives have been as follows:

	1910	1911	1912	1913
Raising cost per foot.....	\$2.95	\$2.31	\$1.88	\$2.22

The record short raise (of about 50 ft. in length) for 1913 had a cost of \$1.65 per foot, and the record long raise (about 100 ft. long) had a cost of \$2.09 per foot, with explosive costs, respectively, of \$0.77 and \$0.99 per foot of raise.

Stoping.—In 1909, when about 74 per cent. of those drills placing holes in the solid orebody were of the reciprocating type of 3-in. piston diameter, the ore production averaged about 20 net tons of ore broken from the solid per 10-hr. drill shift, with an equivalent of 1.1 sticks of 1 by 8 in. of 50 per cent. dynamite per ton of ore.

In 1910, when about 72 per cent. of the producing drills were air-feed stoping (hammer) drills, the tonnage per drill shift rose to 38 net tons with about the same amount of explosive (which cost \$0.055 per net ton of ore broken), and 13.2 tons were broken per 10-hr. shift of men

working in stopes, or 1.32 tons per hour. Although there is no record of the breaking labor prior to this year, the fact remains that in the actual running of the drills only one man was used with a hammer drill while two men were employed with each reciprocating drill.

In 1911, when the hammer drills were about 80 per cent. of the total, the stoping efficiency profited by the improvements in the drilling speed of the hammer drills, and 121 net tons were broken from the solid per drill-shift with about 0.8 stick of 50 per cent. 1 by 8 in. dynamite per ton (at an explosive cost of \$0.041 per ton), and 2.03 net tons were broken per man-hour of men working in stopes.

In 1912, when about 98 per cent. of the stoping drills were hammer drills, 195 net tons were broken per drill-shift with about 0.87 stick of 50 per cent. 1 by 8-in. dynamite per ton (at an explosive cost of \$0.045 per ton), and at the rate of 2.56 net tons per man-hour in the stopes.

In 1913, when all the stoping drills were hammer drills, the length of the working shift was reduced from 10 to 8 hr. in the middle of the year and the tonnage broken per drill-shift fell proportionately to 170 net tons, but remained at approximately the same hourly rating as for the year 1912. However, the tonnage broken per man-shift in the stopes increased slightly to 26.7 net tons (at an explosive cost of \$0.049 per ton), with the consumption of 0.89 stick of 50 per cent. 1 by 8-in. dynamite per ton. The tonnage broken per man-hour was 2.97, which showed a steady gain over previous years.

It should be noted that the explosives charged against stoping include those used by the trammers in blasting ore in the chutes, and thus represent all the dynamite necessary to reduce the ore to the proper size for being handled through chutes and in the mill.

Opencut.—In order to provide broken rock for filling material to fill empty stopes to support the remaining orebody, "mill-holes" are developed in limestone country rock at the surface. For some years it was the practice to use 30-ft. bench-holes in the opencut for quarrying the rock, both 3-in. and 3½-in. reciprocating rock drills being used. It took steady work for two men to sink one 30-ft. hole in a 10-hr. shift, and their work was hazardous because of the inconvenient localities where set-ups were made, and because of the clumsy weight of their machines and the long, heavy drill steels which were handled. After the success of hammer drills in the underground mining operations, they were tried in the opencut work in 1912. Small holes were drilled to an average depth of 16 ft., and were given lighter burdens than had previously been the practice, for the object was to distribute the dynamite more evenly in the rock, as contrasted to churn-drill or mammoth blasts. In the tough, crystalline Franklin limestone this application of hammer drills to quarrying has proved superior to the heavy or mammoth blasts, for the same tonnage can be produced from a bench

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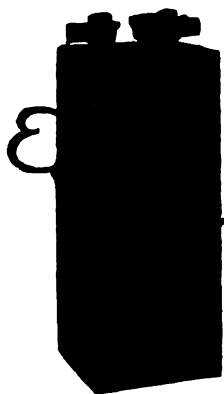
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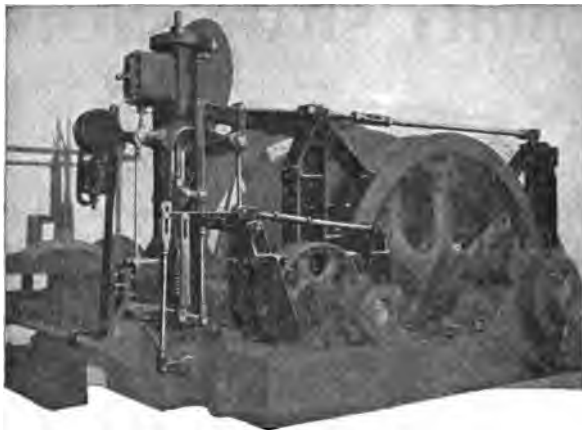
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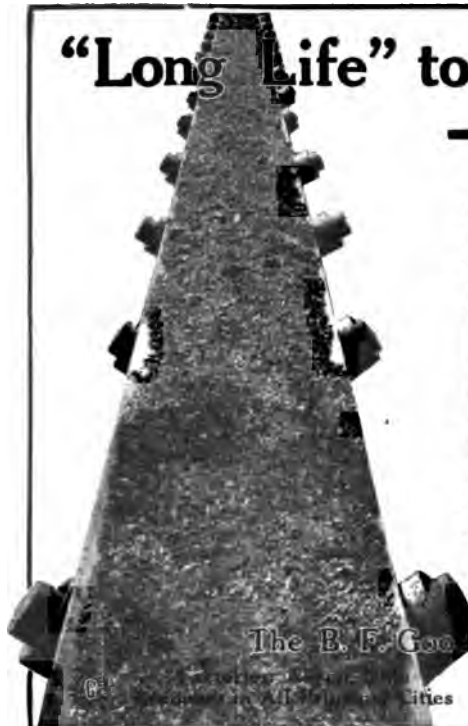
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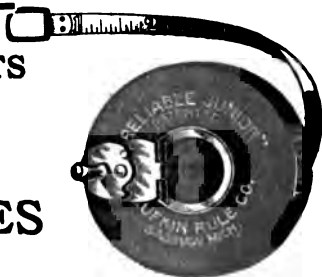
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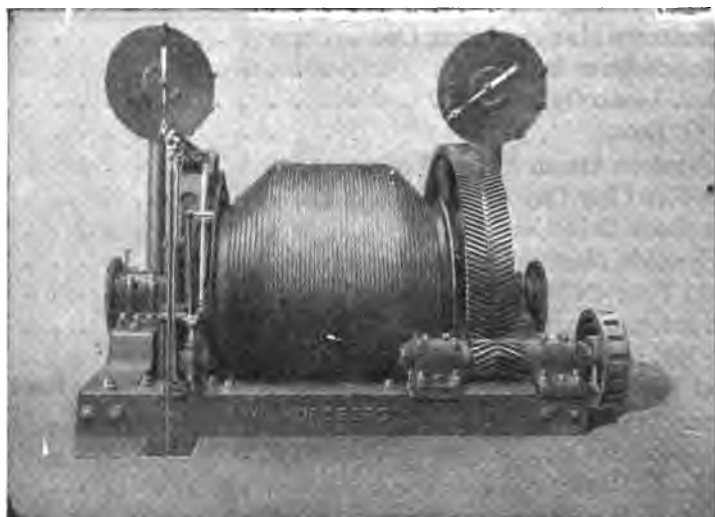
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MARCH, 1915



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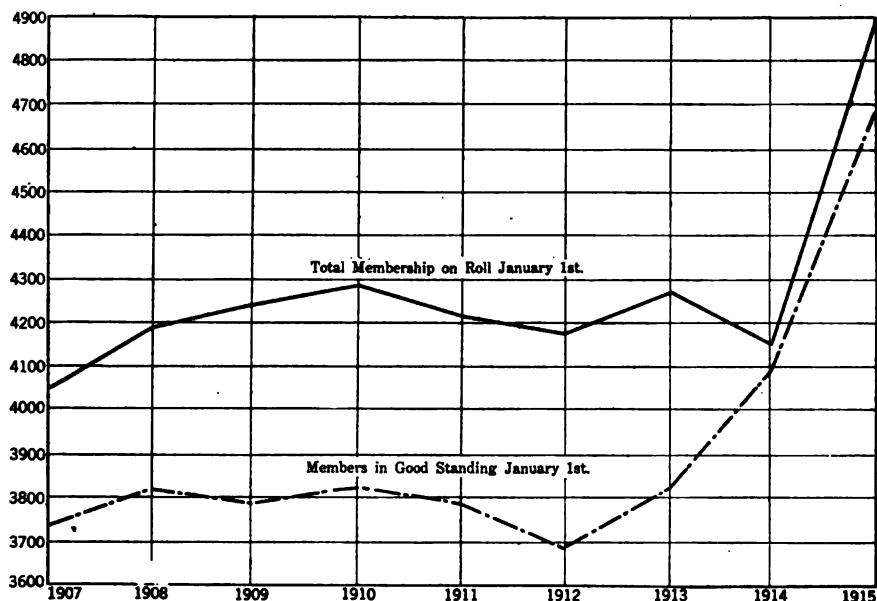
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Growth of Institute Membership

The diagram showing the rate of growth of the Institute printed in the *Bulletin* of December, 1914, was somewhat misleading in that it showed only the total membership on the roll. At the beginning of the year 1914, the new Constitution went into effect, which resulted in the dropping of a large number of men for non-payment of dues. The lower curve in the



accompanying diagram shows that excellent growth was made during 1913 and 1914, the number of members in good standing having increased from 3,820 to 4,700.

Committee on Increase of Membership,
ADOLPHE E. BORIE, Chairman,
THOMAS T. READ, Secretary.



PROPOSAL FOR MEMBERSHIP

[illegible]

[illegible]

ARTICLE II.—MEMBERS.

Institute shall comprise four cl

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Harvard College Library
Aug. 14, 1916.
Bequest of
Erasmus Darwin Leavitt.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 99

MARCH

1915

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Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

SAN FRANCISCO MEETING

The 111th meeting of the Institute for the presentation and discussion of technical papers will be held in San Francisco, Cal., Sept. 16 to 18, 1915. A special train for members of this Institute and for the members of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the Society of Naval Architects and Marine Engineers, and the International Engineering Congress, will be run from New York City to San Francisco, leaving New York on Sept. 9, stopping at Niagara Falls, Colorado Springs and the Grand Canyon, and reaching San Francisco on the evening of Sept. 15. Meetings of our sister societies will be held at approximately the same time as our own in San Francisco and the meeting of the International Engineering Congress will open on Monday, Sept. 20.

All papers to be presented at this meeting of the Institute must be in the hands of the Secretary on or before June 1, 1915.

BOARD OF DIRECTORS

Meeting of Jan. 22, 1915.—Dr. Albert R. Ledoux was unanimously nominated by the Board as one of the representatives of this Institute upon the Engineering Foundation Board.

The following Tellers to count the votes for officers and directors on Feb. 16, 1915, were appointed: David H. Browne, Arthur S. Dwight, and Bradley Stoughton; and the following Tellers to count the ballots on the amendments to the Constitution: Arthur L. Walker, Max Roesler, and Burr A. Robinson.

Karl Eilers was appointed a member of the Library Board of the United Engineering Society, representing this Institute in place of C. R. Corning, resigned.

The Pick and Hammer Club of the University of Oklahoma was established as an Affiliated Student Society.

The sum of \$150 was appropriated to the Columbia Local Section.

The sum of \$29.50 was appropriated to the Chicago Local Section.

The Secretary was authorized to send advance proofs of papers to be published by the Institute to the technical magazines, each proof to be stamped with a release date before which it may not be published.

Upon the recommendation of the Committee on Membership, 46 Members, 3 Associates, 11 Juniors were elected, and the status of one Junior was changed to that of Member.

AFFILIATED STUDENT SOCIETY NOTES

The University of Wisconsin Mining Club meets regularly every two weeks, the program usually comprising talks on mining and geological subjects. The Club is preparing to take a prominent part in the second exposition of the University, to be held Mar. 19 and 20. The officers of the Club for the present school year are: President, Donald Schindler; Vice-President, A. T. Newell; Secretary-Treasurer, R. A. Pallansch; Mucker, Frank Pardee; Assistant Mucker, A. F. Peterson.

The following are the officers of the Mining Association of the University of California for the current semester: President, A. B. Marquand; Vice-President, D. Vedder; Treasurer, O. A. Cavius; Secretary, F. J. Hoenigmann; Librarian, W. G. Farnlacher; Executive Committee, G. H. Smith, E. Woodcock, R. Rhoades.

ANNUAL TABLE OF CONSTANTS

The Institute contributes from its funds to the expense of publication of the *Annual Table of Constants*, and, in consideration of this contribution, members of the Institute are entitled to purchase the *Annual Table of Constants* at a reduced price. Notice of this is hereby given to members, in order that they may claim the reduction when purchasing this volume.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Jan. 10 to Feb. 10, 1915:

N. H. Emmons, 2d, Dover, Mass.
Paul W. Gaebelein, Cornucopia, Ore.
H. D. Kinney, Easton, Pa.
W. D. McDougall, New York, N. Y.
Kirby Thomas, New York, N. Y.
V. T. Lawshe, Newark, N. J.

R. O. Hocking, Redruth, England.
G. H. Carnahan, New York, N. Y.
W. A. Wilson, Salt Lake City, Utah.
H. V. Winchell, Minneapolis, Minn.
Mrs. H. V. Winchell, Minneapolis, Minn.
Keijiro Nishio, Tokyo, Japan.

Henry S. Stebbins, who has been connected with the firm of Oglebay, Norton & Co., Cleveland, since 1892, has resigned to accept a position with **M. A. Hanna & Co.** of the same city.

E. L. Newhouse has been chosen First Vice-President of the American Smelting & Refining Co. in place of the late **Barton Sewell**.

Philip S. Morse, formerly General Manager of Mines, Mills and Smelting Works with the Sulphide Corporation, New South Wales, has resigned that position, owing to poor health, and is back in the United States.

O. M. Kuchs has been promoted from Assistant Superintendent to Superintendent of the International Smelting Works, at Tooele, Utah.

Albert Burch has resigned as General Manager of the Gold Field Consolidated Mines Co., Goldfield, Nev., and **K. M. Simpson** has been appointed Assistant General Manager.

Charles S. Witherell, formerly Superintendent of the Copper Department of the Balbach Smelting & Refining Co., has accepted the position of General Superintendent of the reduction plant of the Chile Exploration Co., Chuchicamata.

Frederick Laist has been made Metallurgical Manager of the Anaconda Copper Mining Co. at Great Falls, Mont., to have general supervision of construction and technical affairs. The local operations will be in charge of **Charles W. Goodale** and **E. P. Mathewson** as heretofore. **Louis V. Bender** becomes General Superintendent of the Washoe plant, with **Harry S. Ware** as assistant.

Eugene Haanel, Director of Mines of Canada, has been elected Vice-President of the Faraday Society of England.

Roswell H. Johnson, Professor of Geology at the University of Pittsburgh, and **L. G. Huntley**, of Pittsburgh, have become associated under the name of **Johnson & Huntley**, and will undertake examinations and reports on oil and gas properties.

Bruce C. Yates, formerly Chief Engineer, has been appointed Assistant Superintendent of the Homestake Mining Co., Lead, S. D.

Frederick K. Brunton is making an examination of a property in Arizona, and will then become connected with the mill of the Portland Gold Mining Co., at Victor, Colo.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

Opening for three unmarried draftsmen with large mining company in Chile, S. A. Two must be designers familiar with the layout and detail work connected with a large milling and smelting plant; the third must be good steel designer. Salary starts day of leaving New York; all expenses paid to Chile and return; three-year agreement required. Applicants should state age, experience, and salary expected. No. 27.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, aged 43, with 25 years' experience as engineer, examiner, and superintendent of steel, copper, lead, nickel, gold mines, and plants. Record for organizing and handling work quickly and economically even in remote districts. Last position five years' heavy reconstruction work covering blast furnaces, converters, roasting and refining plant. Ready to go anywhere, similar fields. Speaks German. No. 188.

Member, aged 36, married, technical graduate, 16 years' experience in coal and ore mines, desires a position as engineer or superintendent, or can fill both positions if mine is not too large. No. 189.

Member, aged 34, technical graduate, with experience in the mining of iron, desires a position as superintendent of a small plant or mine, or assistant to an executive in a large mine. No. 190.

Technical graduate, aged 31, with experience in the metallurgy of copper, is open for engagement. No. 191.

Technical graduate, aged 29, with experience as engineer and safety instructor with coal and steel companies, is desirous of a position as safety instructor or assistant to superintendent of coal mines. No. 192.

Member, technical graduate, practical experience, Elmore flotation process, electrostatic process, and general ore-testing for other flotation and ore-dressing methods. Prefers milling work. No. 193.

Member, with over 15 years' experience as superintendent and assistant general manager in construction work and operating of coal mines and coke plants, desires position in operating department of fuel-mining company. No. 194.

Member, technical graduate, with experience as assayer, cyanide chemist, mill shift boss, mine engineer, and geologist, and as consulting engineer and general manager of a company in Mexico, desires a position, preferably in South America. No. 195.

REPORT OF THE COMMITTEE ON COAL AND COKE FOR THE YEAR 1914

Committee on Coal and Coke

H. M. CHANCE, *Chairman*.

SAMUEL D. WARRINER, *Vice-Chairman*, SAMUEL A. TAYLOR, *Vice-Chairman*

FREDERICK W. C. WHYTE, *Vice-Chairman*.

ELI T. CONNER, *Secretary*, 1315 Stephen Girard Bldg., Philadelphia, Pa.

FRANKLIN BACHE,	GEORGE W. EVANS,	R. V. NORRIS,
SAMUEL W. BEYER,	HENRY S. FLEMING,	THOMAS H. O'BRIEN,
WILLIAM H. BLAUVELT,	FRANK HAAS,	WILLIAM N. PAGE,
FRED M. CHASE,	FRANK A. HILL,	FLOYD W. PARSONS,
THOMAS H. CLAGETT,	CHARLES F. HUBER,	EDWARD W. PARKER,
CLARENCE R. CLAGHORN,	ALBERT B. JESSUP,	EDGAR P. PETTEBONE,
EDWARD H. COXE,	CHARLES E. KREBS,	ERKSINE RAMSAY,
JAMES S. CUNNINGHAM,	W. W. KEEFER,	GEORGE S. RICE,
FRANK W. DEWOLF,	EDWIN LUDLOW,	W. J. RICHARDS,
A. W. DICKINSON,	EVERETT B. MOORE,	CARL SCHOLZ,
HOWARD N. EAVENSON,	MARSHALL G. MOORE,	HARRY H. STOEK,
CHARLES ENZIAN,	ROBERT H. MORRIS,	MORRIS WILLIAMS.

The work of the Committee during the year 1914 was especially directed to the securing of a larger representation in the Institute of those engaged in coal mining and coking. While these industries exceed in volume and in monetary importance all other mining industries in the United States, the membership of the Institute representing them has in recent years been relatively small. It was thought that an increase in the membership of the Institute obtained from among those engaged in these industries was desirable in order to increase interest in the work of the Institute, for this result can only be reached by securing the presentation and publication of a larger number of papers relating to these industries. It is evident that the reading of such papers will tend to increase the attendance at the local and general meetings of the Institute.

Sessions of the Coal and Coke Committee were held and a number of papers relating to these subjects were presented at the meetings at Butte, Salt Lake, and Pittsburgh, the Pittsburgh meeting especially being planned with this in view.

The activities of the Local Section of the Institute organized in the Pennsylvania Anthracite region have reawakened interest in that district and it is hoped that the same object can be obtained in the bituminous coal fields, either through the organization of one or more local sections, affiliation with some of the local coal-mining societies, or through activities of the Coal and Coke Committee.

Referring to the work of the Committee in its efforts to increase the membership of the Institute, the Committee desires to acknowledge the services rendered by many members scattered throughout the various coal-mining districts of the United States, in compiling lists of those qualified for membership in the Institute, but who have not yet become members. The lists so compiled comprised about 1,100 names, which were furnished to the Committee by the co-operation of 40 or 50 members of the Institute.

These lists were carefully culled and revised by correspondence with many members, and from them a list comprising about 568 names was prepared. To each of the persons in this list the Coal and Coke Committee sent an invitation to make application for membership. Perhaps

because these were circular letters and were not of a sufficiently personal character to command attention, a relatively small number of acknowledgments (about 80) were received, of which about 55 were accompanied by applications for membership.

It was then determined to follow this up by correspondence with some 488 from whom no reply had been received.

As a result of this second canvass about 277 replies were received, or 56 per cent. of the number to whom letters were addressed.

These replies are classified as follows:

Deceased.....	1
Addresses defective—could not be found.....	5
Away from home.....	6
Claiming that former communication had not been received..	67
Declining the invitation to make application for membership..	114
Asking for further information.....	6
Stating the matter is held under advisement.....	32
Stating that application would be filed later.....	10
Stating that application has already been filed.....	4
Filing applications.....	28

The total number of applications received to date is about 83, which will represent a quite respectable addition to the membership of the Institute, coming as it does solely from among those engaged in these two industries. It is also quite probable that a number of additional applications were received, as a result of the work of the Committee, these being applications made either through members of the Institute with whom the applicant was personally acquainted, or by transmitting the papers directly to the Secretary. In either case such applications might not appear upon the records of the Coal and Coke Committee. Doubtless a considerable number of the 32 who have the matter under advisement, and of the 10 who state they will file applications later, will ultimately become members of the Institute, so that it is reasonable to assume that the work of the Committee should be credited with about 100 applications.

The conduct of a campaign for the increase in the membership of the Institute from a single center, is not perhaps the best way in which such a canvass could be made. Better results could doubtless be obtained by a systematized effort to awaken the personal interest of members of the Institute who are located in the different coal-mining districts and who are in personal and daily contact with engineers and mine managers qualified for membership. It is thought that such a plan might well be adopted in the near future. The Committee has found that members of the Institute are quite willing to assist the Institute and the Committee in this way, and many of the applications received for membership by the Committee have been obtained either directly or indirectly through the influence of such members.

A very large percentage of the members of the Committee have taken a keen personal interest in the work, and many members of the Institute, who were asked to assist the Committee, rendered most valuable service. The committee feels that the Institute is to be congratulated upon these evidences of the loyal support of its members in all of the different coal-mining districts, without which the work of the Committee could not have been successfully conducted.

H. M. CHANCE, *Chairman.*

REPORT OF COMMITTEE ON PRECIOUS AND BASE METALS

*Committee on Precious and Base Metals*CHARLES W. GOODALE, *Chairman.*L. D. RICKETTS, *Vice-Chairman.*ROBERT C. GEMMELL, *Vice-Chairman.*DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.

SUB-COMMITTEES

COPPER

E. P. MATHEWSON, *Chairman.*

W. H. ALDRIDGE,
W. LAWRENCE AUSTIN,
FREDERICK I. CAIRNS,
DAVID COLE,
FRED W. DENTON,

JOHN C. GREENWAY,
LAFAYETTE HANCHETT,
WILLIAM H. HOWARD,
FREDERICK LAIST,
C. B. LAKENAN,

L. D. RICKETTS,
FOREST RUTHERFORD,
ARCHER E. WHEELER,
A. E. WIGGIN.

GOLD AND SILVER

F. LYNWOOD GARRISON, *Chairman.*

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F. L. BOSQUI,
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CHARLES BUTTERS,
G. H. CLEVINGER,

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JOHN V. N. DORR,
FRANKLIN GUITERMAN,
JAMES W. MALCOLMSON,
CHARLES W. MERRILL,

WILLET G. MILLER,
CHARLES H. MUNRO,
HENRY C. PERKINS,
R. M. RAYMOND,
WHITMAN SYMMES.

LEAD

H. O. HOFMAN, *Chairman.*

LEONARD S. AUSTIN,
O. M. BILHARZ,
JOHN S. CARNAHAN,
ARTHUR S. DWIGHT,

KARL EILERS,
H. A. GUESS,
SIDNEY J. JENNINGS,
FRANK M. SMITH,

ARTHUR THACHER,
BULKLEY WELLS,
RUSH J. WHITE,
WILLIAM WRAITH.

ZINC

GEORGE C. STONE, *Chairman.*

S. E. BRETHERTON,
GELASIO CAETANI,

W. MCA. JOHNSON,
DORSEY A. LYON,
H. A. WHEELER.

A. L. QUENEAU,
C. E. SIEBENTHAL,

MISCELLANEOUS METALS

CHARLES H. FULTON, *Chairman.*

DAVID H. BROWNE,
SIEGFRIED FISCHER,
FRANK L. HESS,

ROBERT M. KEENEY,
GEORGE A. PACKARD,
WALTER M. STEIN,

JOSEPH STRUTHERS,
WILLIS R. WHITNEY.

BOARD OF DIRECTORS,

AMERICAN INSTITUTE OF MINING ENGINEERS:

I submit you herewith a statement covering the activities of your Committee on Precious and Base Metals during the past fiscal year:

Papers Secured and Published in the Transactions

Sub-Committee on Copper, 16 papers.

Sub-Committee on Lead, 10 papers.

Sub-Committee on Zinc, 2 papers.

Sub-Committee on Gold and Silver, 6 papers.

Sub-Committee on Minor Metals, no papers.

Some additional papers were secured but were not accepted by the Publications Committee.

About 25 papers have been promised the various sub-committees and are in course of preparation.

Papers on certain metallurgical subjects have not been secured for the following reasons:

1. Unwillingness of members familiar with the subject to devote the time necessary to prepare the paper.
2. Unwillingness to make public the information because of (a) competition, (b) litigation.
3. Fear that the data are not complete enough.
4. Previous agreements to publish the papers in other channels.

Commenting on these reasons, I am of the opinion that reason number one is seldom justifiable. It is usually nothing but inexcusable selfishness. These men freely avail themselves of the information furnished by their less selfish associates but are unwilling to give anything in return.

Reason number two is often sufficient. Where processes are in litigation it appears inadvisable to furnish certain kinds of data concerning them. Among the minor metals, such as aluminum, cobalt, tungsten, etc., there seem to be many details of practice which it would be inadvisable to disclose to competitors.

Reason number three is seldom justified. If we all waited until the last thought has been applied before publishing, our ideas would die with us. By publishing data as they are acquired they become available to other experimenters promptly, often saving much time and useless repetition.

Reason number four is a difficult one to handle. The technical press competes for desirable papers and offers pecuniary and other inducements to authors. Our only province in this matter is to remind authors that their writings are not usually so well preserved or so available for future reference in the technical magazines as they are in the regularly bound volumes of the *Transactions* of the Institute. The technical magazines have the privilege of republishing Institute articles, while it is not the policy of the Institute to publish articles which have appeared elsewhere previously. Separate bound copies of a paper are easily secured in Institute publications; this is sometimes not feasible in the technical magazines.

What progress the Previous and Base Metals Committee has made during the past year has been made possible by the willing and active support tendered the Chairman by the members.

Respectfully submitted,

C. W. GOODALE, *Chairman*.

COLORADO LOCAL SECTION

Executive Committee

CHARLES A. CHASE, *Chairman*

S. A. IONIDES, *Vice-Chairman*

C. LORIMER COLBURN, *Secretary-Treasure*, 614 Ideal Bldg., Denver, Colo.

F. H. BOSTWICK,

W. G. SWART.

The annual meeting of the Colorado Local Section of the Institute was held at the Savoy Hotel, Denver, Jan. 15. The Chairman, Frank Bulkley, presided and made a short address on the work of the Section. The following officers were elected:

Chairman, Charles A. Chase.
Vice-Chairman, S. A. Ionides.
Secretary-Treasurer, C. Lorimer Colburn.
Directors, F. H. Bostwick, W. G. Swart.

Dr. R. B. Moore, of the U. S. Bureau of Mines, delivered a very instructive and entertaining lecture on The Rare Gases of the Atmosphere. The lecture was illustrated by charts and Geissler tubes showing the brilliant colors of the gases when subjected to high-tension electrical influence.

JAMES M. McCLAVE, *Acting Secretary*.

MONTANA LOCAL SECTION

Executive Committee

FRANK M. SMITH, *Chairman*
JAMES L. BRUCE, *Vice-Chairman*

D. C. BARD, *Secretary-Treasurer*, Montana State School of Mines,
Butte, Mont.

FREDERICK LAIST

W. C. SIDERFIN

The annual meeting of the Montana Local Section of the Institute was held at the Silver Bow Club, Butte, Feb. 5, at which papers as follows were read:

The Concentrator of the Timber Butte Milling Co., by Theodore Simons. The Boulder Batholith of Montana, by Paul Billingsley.

D. C. BARD, *Secretary*.

BOSTON LOCAL SECTION

Executive Committee

HENRY L. SMYTH, *Chairman*
ALFRED C. LANE, *Vice-Chairman*

AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.

ROBERT H. RICHARDS

ALBERT SAUVEUR

Meets on the First Monday evening of each winter month.

Twenty-fifth Meeting

The Twenty-fifth meeting of the Boston Local Section of the Institute was held at the University Club, Boston, on Monday, Feb. 1. The Chairman, Henry L. Smyth, presided, and 15 members and guests were present. The Secretary of the Institute made a brief address, after which R. V. Norris presented an illustrated paper on The Valuation of Coal Lands. An extended discussion on the subject followed.

AUGUSTUS H. EUSTIS, *Secretary*.

SAN FRANCISCO LOCAL SECTION*Executive Committee*G. HOWELL CLEVENGER, *Chairman*C. W. MERRILL, *Vice-Chairman*JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto. Cal.

F. W. BRADLEY

ANDREW C. LAWSON

Meets on the second Tuesday of each month.

A meeting of the San Francisco Local Section was held on Feb. 9, at the Clift Hotel, San Francisco. The speaker of the evening was C. E. Wells, Chief Metallurgist and Chemist to the Selby Smoke Commission, who presented an abstract of the report of the Commission, illustrated by lantern slides.

JAMES C. RAY, *Secretary*.

THE ENGINEERING FOUNDATION

On Wednesday evening, Jan. 27, 1915, were held in the auditorium of the Engineering Societies Building the inaugural exercises of The Engineering Foundation, established by the United Engineering Society. The initial gift to the Foundation of \$200,000, made by Ambrose Swasey, Past President of the American Society of Mechanical Engineers, was formally accepted at this time. So far as known this is the first instance of a foundation devoted to engineering purposes. This meeting constituted also an appropriate tribute to the generous gift made to the Foundation and gave opportunity for expression of the satisfaction and approval which are felt by engineers everywhere at the establishment of a means of promoting the good of mankind through the work of the engineer along the broadest lines.

Gano Dunn, President of the United Engineering Society and Past President of the American Institute of Electrical Engineers, acted as presiding officer, and on the platform were seated representatives of the three societies constituting the United Engineering Society, and of the American Society of Civil Engineers. Members of the four societies and many other friends interested in the profession composed the audience.

Address by Gano Dunn

The first address of the evening was made by Gano Dunn, who reviewed the history of the Engineering Societies Building, the purpose of the United Engineering Society, and the formation of The Engineering Foundation.

Mr. Dunn stated that we have foundations of many kinds,—for philanthropy, for civic betterment; foundations for many noble purposes seem to characterize American public life. But as yet we have to hear of a foundation in engineering. At last our turn has come, at last the human element is recognized among engineers in benefaction, and at last engineering is included among those human activities which are felt to be sufficiently important to promote.

The speaker explained that when the trustees of the United Engineering Society came to consider all the possible objects and uses to which a

foundation could be put, they felt it could be best administered if in the hands of a responsible and distinguished board, separate from their own, with entire freedom as to the expenditure of their resources. It was felt further that the work could never be successful or representative of the whole field of engineering if its roots were in only the three national engineering societies which constitute the United Engineering Society. It was felt that the American Society of Civil Engineers must take part in the work of the United Engineering Society, or the work of the Foundation could not be expected to be successful. The structure of the Engineering Foundation, therefore, was so arranged that it should be managed by a Board of eleven members, of which the President of the United Engineering Society should be *ex officio* one, two members from each of the founder societies, two from the American Society of Civil Engineers, and two from the world at large. In this latter provision the trustees hoped to introduce into the Foundation an outside element and a point of view that would forever save the Board from becoming narrow, and would likewise inspire new ideas. The speaker said that in his opinion they never did so wisely as when they called upon the world in general to share with them in administering this great trust.

Mr. Dunn said that in proposing this structure to our sister society and inviting them to share in the management of the trust, a letter of acceptance was received which the trustees felt insured that the future of The Engineering Foundation would be a brilliant one.

Mr. Dunn, in announcing the donor, said that Mr. Swasey had requested that his part be made as small as possible, but that we could never forget how large is the part he played not only in giving this endowment, but in initiating and creating the structure that undoubtedly will be the receptacle of many future endowments, and the center and source of much important work in the future.

Remarks by Ambrose Swasey

Mr. Swasey was then called upon by the Chairman and very briefly expressed his appreciation of the cordial manner in which the announcement had been received. He believed that the meeting was also evidence of the appreciation felt of what had been accomplished by the United Engineering Society and the work which they had done in planning and bringing about this Foundation. It is a very simple matter to throw a few shovelfuls of coal under the boiler, he said. We must have fuel if we would have power. But to the engineers who designed the boiler and the engine, and who so proportioned them and so correlated them that the products of combustion can be used to greater advantage than ever before for the benefit of mankind, the greater credit belongs.

Address by Robert W. Hunt

Robert W. Hunt, Past President of the American Institute of Mining Engineers, paid a personal tribute to Mr. Swasey, as a lifelong friend, and said that it is one of the few certainties of this world that the greatest earthly happiness comes to those who have made others happy. The United Engineering Society, which will administer this Foundation "to develop, direct and conserve the forces of nature for the benefit and uplift of mankind through the instrumentality of the engineer," was made possible by the generosity of one man, Andrew Carnegie.

This Foundation also is the conception of one man who does not seek individual aggrandizement, but rather through his own munificence to create a base upon which others can unite with him in building a structure which will benefit all humanity. This structure will not be his nor any other man's, but to each one who may hereafter add to its growth and usefulness will be given the happiness incident to helping to uplift mankind. This is no new rôle to Mr. Swasey, to whom all human beings, whether they have the yellow skin of the Far East or are the laboring boys and men of his own country, are fellow-mortals. While he built telescopes to comprehend in part the incomprehensible universe, his love for and his desire to serve his fellow men has grown and has manifested itself in many ways.

Address by Henry S. Pritchett

Dr. Henry S. Pritchett, President of the Carnegie Foundation for the Advancement of Teaching, spoke upon the significance of the gift and its meaning for engineering science and civilization. Great numbers of research institutes already exist in Europe and America, but never has such an agency for research been placed in the hands of a group of engineering societies. The word research has become almost a shibboleth in university circles; but it must be confessed that a great proportion of this is the imitation, not the reality. In the sense in which it is used in this Foundation, he thought, it means the attempt to formulate and solve those problems of science which touch most directly the comfort, the happiness and the progress of the nation and of the world. In this process it requires quite as high an order of ability to formulate the problem clearly as to solve it.

"Research considered as a process of scientific thinking and discovery is not a task to be set before the immature and the untrained. It is quite true that the best training for research may not always lie in university halls. A long roll of illustrious men whose names the world cherishes found their way to the solution of the great problems of science and of industry by other paths than the university. This means simply that they became thinkers in their own way.

"The process of research, therefore, as one may expect it to be carried forward by such an agency as this, involves first the clear formulation of problems, and then the attack upon the problem by all the avenues and by all the agencies of science, and this process, it may well be assumed, will be successfully carried out only by those men who have learned to think, who are familiar already with the present state of science, and who—using this as a basis—push forward the boundaries of our knowledge into new fields. True research implies clear thinking, minute knowledge of the field of science, and patient endeavor.

"The development of such a research agency by the great engineering societies, through a fund entrusted to them, has, to my thinking, far reaching significance, first, for engineers themselves; secondly, for research, and in the third place, for the public, the great body of citizens.

"So far as engineers themselves are concerned, the offer and acceptance of this foundation signalizes the fact that engineering in America is to-day a profession, not a business. For two centuries and more the civilized nations have recognized three great professions—that of the law, of medicine, and of the clergy. Only within our generation has engineering been admitted to the fellowship of this group; and it has come because the world now recognizes that the man who serves in the field of applied science, if he serve in the right spirit, gives a part of his energy to the service of the public. It is in just this respect that business differs from a profession. Physician and lawyer and clergyman accept pay, but in each of these professions it is clearly recognized that the service must not be governed by the pay, that a large part of the energy of the profession must be given to the service of the common good, that members of a profession occupy a quasi-public relation. Only when the men of a calling attain this sense of professional consciousness can the calling pass from that of a business to that of a profession. The members of a vocation who take upon themselves the responsibil-

ity for the direction and encouragement of scientific research have risen out of the business of engineering into the profession of engineering.

"With regard to research it may be said that, while engineering research in its essence partakes of the nature of all research, it undertakes to formulate an answer to certain questions which are not likely to be dealt with successfully elsewhere. Ever since universities and institutes for research have existed there has gone on the discussion between what is called theory and what is called practice. As Professor Pupin has well put it, 'It is the practice which formulates the question; theory seeks to answer it.'

"It is the man in the practice who formulates the question. It is the scientist who finds the answer.

"The criticism of university research has been that theory and practice, the man who formulates and the man who solves, have not always been happily related. In a research agency conducted by associations of engineers there ought to be the most fortunate alliance of theory and practice. Here, the practitioner who questions and the man who seeks to solve, the man who formulates and the man who solves, ought to find their closest association. And it will be interesting to observe how, under such auspices, theory and practice will learn to go hand in hand.

"To the public the foundation of such a research agency in the field of applied science may well mean the inauguration of a series of studies having the greatest significance for public well-being and for public needs. The most striking illustration of such a result has been already given in the history of a small research laboratory of applied science, started in the technical school, at Charlottenburg, some forty years ago. The effort arose out of the questions formulated and laid before the scientific staff of the technical school by the men of the steel industry; and out of the solutions of these questions the steel industry of Germany grew. Gradually the inquiry arose, why should there not be an institute of engineering research to which any citizen engaged in the industries may bring his problems? to whose laboratories may be brought not alone the questions relating to the manufacture of steel, but those relating to the chemical industry, textile industry, electrical manufacturing and industries of every sort, in a word, all the questions which to-day in the manufacturing establishments of a great nation are constantly being asked?

"Out of the little research laboratory at Charlottenburg have grown the great testing laboratories (Versuchsanstalt) covering many acres and offering themselves freely to the solution of every technical question that the industries in Germany can present. Any company engaged in the industries may bring its problems here. Any experts may come in and take part in the study, all the literature on the subject is made available, and under the combined efforts of the expert of the government laboratory and the experts of the industry a solution is generally found.

"The process is illustrated by the story which a paper manufacturer in Berlin told to me some years ago. His company manufactured wood pulp from the forest of a certain region. A change was made and their wood supply was drawn from another region. The formulae they had hitherto used no longer sufficed, and despite every effort of their own no solution had been found. Financial ruin stared the company in the face. The problem was carried up to the government laboratories of engineering research. At the end of two months all difficulties had vanished and prosperity returned to the manufacturer.

"Dr. Martens, the director of the research institute at Grosslichterfelde, told me that the complete and orderly collection of scientific literature and of scattered scientific solutions in research made available at Grosslichterfelde was one of the great factors in the success of the research establishment. 'Four times out of five,' said he, 'when a manufacturer brings us a problem we find that it has already been solved somewhere in the world by somebody, but by someone who generally used the solution in a different connection. Our greatest service is to place at the disposal of the inquirer the entire literature of his subject, and in this he will generally find a solution of his question already made.'

"Who can tell but that this modest agency for research, under the direction of the great engineering societies of America, may develop in due course into a national establishment to which any man in the industries may bring his problem with full hope of solution?

"We found here to-night an agency for human uplift and human development. It is founded not in brick and mortar but in the living engineers of these great engineering societies, which shall go on from decade to decade and from generation to generation. If our civilization endures, this association of engineers will develop with the ages. They will never die. And in their hands this agency of research will also be immortal, serving humanity from century to century and from age to age.

"It has been founded by an engineer at once wise and modest. To have founded such an immortal agency for human endeavor is to become oneself a partaker in immortality."

Remarks by Charles McDonald

Following the address by Dr. Pritchett the Chairman called upon Charles McDonald, Past President of the American Society of Civil Engineers, saying that if The Engineering Foundation had no other virtue than to bring into closer unity the four great engineering societies, it would have accomplished results of greatest value.

In response Mr. McDonald said that the American Society of Civil Engineers welcomed the efforts to be made through this gracious gift to The Engineering Foundation. His society would be glad to co-operate in working for the general good of the profession. No matter what organization he was representing, he wished to be considered at this time as a member broadly of the whole profession.

Remarks by Alexander C. Humphreys.

Dr. Alexander C. Humphreys, Past President of The American Society of Mechanical Engineers, then formally accepted Mr. Swasey's gift in behalf of the Board of Trustees of the United Engineering Society.

The acceptance of the fund, he said, meant the acceptance of a grave responsibility, which would, he believed, be met with the full determination on the part of the representatives, present and future, of the four societies to administer this fund and additional contributions in the interest of the larger needs of mankind. He hoped that the Foundation in its activities would always stand for the truth, the whole truth and nothing but the truth.

AMBROSE SWASEY

Ambrose Swasey, the donor of the initial fund for The Engineering Foundation, is widely known as a member of the firm of Warner & Swasey of Cleveland, Ohio, prominent machine tool builders and the foremost builders of telescopes in the world. Mr. Swasey is known also, in addition to his engineering achievements, for his practical efforts toward scientific education and the advancement of the profession. His gift for the establishment of The Engineering Foundation is in line with these undertakings, which may be destined to outlast his fame as an engineer. Mr. Swasey gave the handsome observatory to Denison University at Granville, Ohio, and the science building for the University of Nanking and the Young Men's Christian Association Building for the Canton Christian College now being erected in China, were made possible through his gifts. He has also, as president of the Warner & Swasey Company, interested himself earnestly in the establishment and conduct of its school of apprentices, and indeed the weight of his influence is to be found in every project tending to the development and encouragement of the human element, helping men to help themselves.

DINNER TO DR. F. G. COTTRELL

Dr. Frederick G. Cottrell was given a dinner at the Hotel Plaza, New York, on the evening of January 15, in recognition of his achievement in

the development of the process for electrical fume precipitation and, in particular, of his munificence in transferring the patent rights to this invention to the Research Corporation, by which procedure funds were made available for further scientific research. The dinner was given under the auspices of the Mining and Metallurgical Society of America, the American Electrochemical Society, and the American Institute of Mining Engineers, Dr. George F. Kunz being Chairman of the Committee on Arrangements.

Sidney J. Jennings presided, and speeches were delivered by Dr. Charles D. Walcott, Director of the Smithsonian Institution, representing the Research Corporation; W. R. Ingalls, representing the Mining and Metallurgical Society of America; F. A. Lidbury, representing the American Electrochemical Society; and William L. Saunders, representing the American Institute of Mining Engineers. The high regard in which Dr. Cottrell is held by reason of his scientific attainments and public spirited actions was voiced by all of the speakers. Dr. Cottrell in reply modestly stated that he was but one of the donors of the patents to the Research Corporation since his associates and co-workers, Dr. Harry East Miller, E. S. Heller, and Prof. Edmond O'Neill, were also highly instrumental in the development of the Cottrell process and were thus entitled to credit as donors.

A minute adopted at the Annual Meeting of the Research Corporation, Jan. 15, 1915, contains an acknowledgment of the debt owed to Dr. Cottrell, as well as a brief historical statement, showing the events which culminated in the formation of the Research Corporation. This minute follows:

In 1911 Frederick G. Cottrell, B. S., Ph. D., offered to transfer to the Smithsonian Institution substantially all his rights in his inventions and patents covering the processes known as the electrical precipitation of suspended particles in order that the profits resulting from the application of the patents, already well assured, might be applied to the advancement of scientific research and investigation. Naturally this proposition was at once recognized as both generous in spirit and highly original in conception. It was Dr. Cottrell's ideal to render discovery already made the mother of new discovery, and thus contribute to the scientific and technical development of the industrial arts.

This far-sighted and patriotic conception found its realization through the "Research Corporation," which for administrative reasons was substituted for the Smithsonian Institution as the custodian of Dr. Cottrell's endowment. The objects of the Research Corporation as stated in its charter are:

"To provide means for the advancement and extension of technical and scientific investigation, research and experimentation by contributing the net earnings of the corporation, over and above such sum or sums as may be reserved or retained and held as an endowment fund or working capital, to the Smithsonian Institution, and such other scientific and educational institutions and societies as the board of directors may from time to time select in order to enable such institutions and societies to conduct such investigations, research and experimentation."

Organized in 1912 as a stock corporation, but precluded by its charter from paying dividends and capitalized by a group of gentlemen desirous of furthering Dr. Cottrell's objects, without personal profit, the Research Corporation undertook and successfully accomplished the installation of the Cottrell processes in various industries throughout the country, with the result that in two years' operation its surplus has provided the capital of \$20,000 required by its charter, and a fund of over \$100,000 for scientific research.

The accomplishment of this remarkable result, which both justifies Dr. Cottrell's expectations and realizes his hopes, is due to his generosity and foresight, and the directors of the Research Corporation take this occasion to express to Dr. Cottrell their appreciation of his contribution to scientific achievement and to the advancement of science, and tender him their cordial congratulations and sincere thanks for the public service which he has rendered.

EMMONS VOLUME ON ORE-DEPOSITS

A continuation of the "Posepny" Volume

Comprising papers descriptive of ore-deposits and discussions of their origin, edited, with an introduction, by Dr. S. F. Emmons. The volume contains also a Biographical Notice of Dr. Emmons by his associate and friend, Dr. George F. Becker, and a comprehensive Bibliographical Index of the Science of Ore-Deposits, prepared by Prof. John D. Irving, of the Sheffield Scientific School of Yale University. Dr. Emmons had finished his editorial work and written his Introduction before his lamented death in 1910; and the Volume contains his last words upon the subject to which he had given the work of his life, and on which he was justly regarded as the foremost authority.

Contents

- Genesis of Certain Ore-Deposits. By S. F. EMMONS.
 Structural Relations of Ore-Deposits. By S. F. EMMONS.
 Geological Distribution of the Useful Metals in the United States. By S. F. EMMONS. Discussion, by JOHN A. CHURCH, ARTHUR WINSLOW, S. F. EMMONS, and WILLIAM HAMILTON MERRITT.
 Torsional Theory of Joints. By GEORGE F. BECKER. Discussion, by H. M. HOWE, R. W. RAYMOND, C. R. BOYD, and GEORGE F. BECKER.
 Allotropism of Gold. By HENRY LOUIS.
 Superficial Alteration of Ore-Deposits. By R. A. F. PENROSE, JR.
 Some Mines of Rosita and Silver Cliff, Colorado. By S. F. EMMONS.
 Genesis of Certain Auriferous Lodes. By JOHN R. DON. Discussion, by JOSEPH LE CONTE, S. F. EMMONS, G. F. BECKER, ARTHUR WINSLOW, W. P. BLAKE, and J. R. DON.
 Influence of Country-Rock on Mineral Veins. By WALTER HARVEY WEED.
 Igneous Rocks and Circulating Waters as Factors in Ore-Deposition. By J. F. KEMP.
 Consideration of Igneous Rocks and Their Segregation or Differentiation as Related to the Occurrence of Ores. By J. E. SPURR. Discussion, by A. N. WINCHELL.
 Chemistry of Ore-Deposition. By WALTER P. JENNEY. Discussion, by JOHN A. CHURCH.
 Ore-Deposits near Igneous Contacts. By WALTER HARVEY WEED. Discussion, by W. L. AUSTIN.
 Ore-Deposition and Vein-Enrichment by Ascending Hot Waters. By WALTER HARVEY WEED.
 Basaltic Zones as Guides to Ore-Deposits in the Cripple Creek District, Colorado. By E. A. STEVENS.
 Geological Features of the Gold-Production of North America. By W. LINDGREN. Discussion, by W. G. MILLER and W. L. AUSTIN.
 Osmosis as a Factor in Ore-Formation. By HALBERT POWERS GILLETTE.
 Ore-Deposits of Sudbury, Ontario. By CHARLES W. DICKSON.
 Genesis of the Copper-Deposits of Clifton-Morenci, Arizona. By W. LINDGREN.
 Copper-Deposits at San Jose, Tamaulipas, Mexico. By J. F. KEMP.
 Magmatic Origin of Vein-Forming Waters in Southeastern Alaska. By A. C. SPENCER.
 Genetic Relations of the Western Nevada Ores. By J. E. SPURR.
 Are the Quartz Veins of Silver Peak, Nevada, the Result of Magmatic Segregation? By J. B. HASTINGS.
 Occurrence of Stibnite at Steamboat Springs, Nevada. By W. LINDGREN.
 Summary of Lake Superior Geology with Special Reference to Recent Studies of the Iron-Bearing Series. By C. K. LEITE.
 Geological Relations of the Scandinavian Iron-Ores. By H. SJÖGREN.
 Formation and Enrichment of Ore-Bearing Veins. (With Supplementary Paper.) By GEORGE J. BANCROFT.
 Distribution of the Elements in Igneous Rocks. By H. S. WASHINGTON.
 Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits of the United States. By W. H. EMMONS.
 Cognate Papers.
 Bibliography of the Science of Ore-Deposits. By J. D. IRVING, H. D. SMITH, and H. G. FERGUSON

The volume contains 1002 pages. Price, bound in cloth, \$5; in half-morocco, \$6. Both the Emmons and the Posepny Volumes on Ore-Deposits, bound in cloth, \$8; in half-morocco, \$10.

READY FOR DISTRIBUTION

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining

LES POUDRES ET EXPLOSIFS ET LES MESURES DE SECURITE DANS LES MINES DE HOUILLE.

By L. Vennin et G. Chesneau. Paris, 1914.

METAL MINE ACCIDENTS IN THE UNITED STATES DURING 1913. Technical Paper 94, U. S. Bureau of Mines. Washington, 1914.

MINING INDUSTRY IN NORTH CAROLINA DURING 1911 AND 1912. Economic Paper 34, North Carolina Geological Survey. Raleigh, 1914.

MODERN PRACTICE IN MINING. Vol. III. Methods of Working Coal. By R. A. S. Redmayne. London-New York, 1914.

Metallurgy

AUSFUEHRLICHES HANDBUCH DER EISENHUTTENKUNDE. Band I-III, IV, pt. 1-2. Braunschweig, 1891-96, 1902, 1906-08.

- CEMENTATION OF IRON AND STEEL.** By Frederico Giolitti, translated from the Italian by J. W. Richards and C. A. Rouiller. New York, 1915.
- DIE EISENPROBIERKUNST.** By Hermann Wedding. Braunschweig, 1894.
- EINFÜHRUNG IN DIE EISENHÜTTENKUNDE.** Leipzig, 1913.
- ELECTRIC FURNACE IN METALLURGICAL WORK.** Bull. 77, U. S. Bureau of Mines. Washington, 1914.
- GALVANISED IRON, ITS MANUFACTURE AND USES.** By James Davies. London-New York, 1914.
- GENERAL FOUNDRY PRACTICE.** By William Rouxburgh. London, 1914.
- HARDENING AND TEMPERING OF STEEL.** By Fridolin Reiser. London, 1903.
- HÜTTENWESEN.** By W. Borchers. Halle a. S., 1908.
- KURZE EINFÜHRUNG IN DEN INNEREN GEFÜGEAUFBAU DER EISENKOHL-STOFF-LEGIERUNGEN.** By O. Kröhnke. Ed. 2. Berlin, 1911.
- LEAD AND ITS COMPOUNDS.** By Thomas Lambert. London, 1902.
- PROGRESSIVE FURNACE HEATING.** By Alfred G. King. New York, 1914.
- A STUDY OF THE MALLEABLE FURNACE.** Harbison-Walker Refractories Co. Pittsburgh, 1915. (Gift of company.)

[NOTE.—This is the third of the little volumes published by the Harbison-Walker Refractories Co., of which the first is entitled, "A Study of the Open Hearth," and the second, "A Study of the Blast Furnace." "A Study of the Malleable Furnace" has these characteristics in common with its two predecessors, that it has condensed into a very small space between its covers a very large amount of technical information of real value and it is entirely free from advertising of any kind throughout the text. All men familiar with the literature of iron and steel are aware of the cordial welcome accorded by technical engineers to the two preceding volumes and we prophesy for this third volume no less hearty appreciation. The comprehensive scope of the book is shown by the contents, which includes a sufficient statement of the characteristics of malleable iron, the process of manufacture, the relative advantages of different furnaces, the design and construction of the air furnace, a study of the operation both from the chemical and metallurgical standpoint, of the annealing of malleable castings, of the microstructure of malleable cast iron, including a brief section on elementary metallography, and finally a record of a typical heat, in which information of great value is included, such as the analysis and proportion of fuel, the contents of the charge, the temperatures of the furnace at different points from beginning to end of the heat, analyses of the slag and metal at different stages of the operation, and a series of photographs of the fractures.]

The book is illustrated by excellent drawings of furnaces, fractures, and micrographs. We commend it to the attention of all those who wish to be well informed on the science of this very interesting material.—B. S.]

Geology, Mineralogy, and Mineral Production

- COLORADO.** Topographic and Geologic Map, 1913. (Gift of Colorado State Geological Survey.)
- DEPOSITS OF THE USEFUL MINERALS AND ROCKS, THEIR ORIGIN, FORM AND CONTENT.** Vol. I. By F. Beyschlag, J. H. L. Vogt, and P. Krusch, translated by S. J. Truscott. London, 1914.
- LEAD AND ZINC DEPOSITS OF NORTHWESTERN ILLINOIS.** Bull. 21, Illinois Geological Survey.
- MINERALS OF PENNSYLVANIA.** Report No. 9. Pennsylvania Topographical and Geologic Survey. Harrisburg, 1913.
- PRODUCTION OF IRON AND STEEL IN CANADA, 1913.** Publication 315. Ottawa, 1914.

Chemistry

- CHEMIE DER ERDE,** Vol. I, pt. 1. By G. Linck. Jena, 1914.
- NOTES ON THE SAMPLING AND ANALYSIS OF COAL.** Technical Paper 76, U. S. Bureau of Mines. Washington, 1914.

Non-Metallic Minerals

- BLACK DIAMOND'S YEAR BOOK, 1914.** Chicago, 1914.

- PETROLEUM AND NATURAL GAS IN OKLAHOMA. By L. C. Snider. Oklahoma City, 1913.
- POTTERY CLAYS OF MISSISSIPPI. Bull. 6, Mississippi State Geological Survey. Jackson, 1914.
- PRODUCTION OF CEMENT, LIME, CLAY PRODUCTS, STONE, AND OTHER STRUCTURAL MATERIALS IN CANADA, 1913. Publication No. 318. Ottawa, 1914.

General

- KONSTRUKTIONSTAHL. Ein praktisches Handbuch über die Festigkeits-Eigenschaften von Stahl und Eisen. By Otto Thallner. Freiberg in Sachsen, 1904.
- ROBERTS-AUSTEN. A Record of His Work. Compiled and edited by S. W. Smith. London, 1914.

Trade Catalogues

- CHICAGO PNEUMATIC TOOL Co., Chicago-New York. Bull. 34-K. Class N-SO and N-SG fuel-oil and gas driven compressors. Dec., 1914.
- CONNERSVILLE BLOWER Co., Connersville, Ind. Blowers, description of various types.
- DEAN BROS. STEAM PUMP WORKS, Indianapolis, Ind. Pony Catalogue No. 88.
- KELLY & JONES Co., Greensburg, Pa. Catalogue M. Illustrated catalogue and price list, 1914.
- GENERAL ELECTRIC Co., Schenectady, N. Y.
- Bull. 48702. Fabroil gears. 1915.
- Bull. 44403. GE-222-G, 600 and 600/1200 volt ventilated commutating pole railway motor. Dec., 1914.
- LESCHEN, A. & SONS ROPE Co., St. Louis, Mo. Freight transportation by the water route. Jan., 1915.
- PELTON WATER WHEEL Co., San Francisco-New York. Pelton Water Wheel, description. 1914.

Notes on Trade Literature

The Jeffrey Manufacturing Co., Columbus, Ohio. Bulletin 143. Contains 19 pages and gives complete information about their standard line of malleable and steel elevator buckets.

Bulletin 167. Contains 24 pages of interesting illustrations and descriptive matter which gives a comprehensive idea of the wide variety and adaptability of their belt conveyor equipments for handling practically all kinds of materials.

Both of these Bulletins will be sent free to interested persons upon request.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Jan. 10 to Feb. 10, 1915:

Members

- AGASSIZ, RUDOLPH LOUIS, Vice-Pres., Calumet & Hecla Mining Co., Boston, Mass.
 AMIDON, RICHARD G., Min. Engr., 3659 Girard Ave. N., Minneapolis, Minn.
 ANDERSON, HECTOR G. S., Min. Engr., 323 N. 13th St., Muskogee, Okla.
 ANDERSON, JOHN CARTER, Petroleum Engr., 2307 S. 33d St., Omaha, Neb.
 CLARKE, ALEXANDER FIELDER, Works Mgr., Firth-Sterling Steel Co., McKeesport, Pa.
 CROSS, CARLOS GARCIA, Min. Engr., Calle Valparaiso 194, Vina del Mar, Chile.
 DAWSON, THOMAS WILLIAM, Min. Engr., Asst. Chief Engr., H. C. Frick Coke Co.,
 Scottsdale, Pa.
 DEHUFF, ARTHUR I., Chem., Metaline Falls Mill, Lehigh Portland Cement Co.,
 Metaline Falls, Wash.
 DIAZ, LUIS F., Engr. of Mines, Apartado 69, Cerro de Pasco, Peru.
 EVANS, SAMUEL, Chairman, Mng. Dir., Crown Mines, Ltd.,
 153 Nugget St., Johannesburg, So. Africa.
 FRENCH, BURR J., Min. Engr., Cinco Minas Co., Magdalena, Jal., Mexico.
 FRIEDRICH, ALFRED KARL, Min. Engr., 3226 W. 14th St., Cleveland, Ohio.
 HARLAND, HENRY LUMLEY, Battery Mgr., Robinson Gold Min. Co.,
 P. O. Box 1024, Johannesburg, So. Africa.
 HILL, BENJAMIN F., 302 N. 4th St., Cripple Creek, Colo.
 INCH, JOSEPH, Min. Engr., Cia. De Minas De Cobre De Gatico, Gatico, Chile.
 JÜSSEN, EDMUND, Min. Engr., 57 Post St., San Francisco, Cal.
 KERR, GUY MANNING, Asst. Supt., Garfield Smelting Co., Garfield, Utah.
 LAWRENCE, HARRY C., Supt., Inter-State Iron Co., Grand Rapids, Minn.
 LIVINGSTON, D. C., Prof. of Min. Engr., Univ. of Idaho, Moscow, Idaho.
 MCHARG, HENRY K., JR., Vice-Pres., Virginia Iron, Coal & Coke Co., Roanoke, Va.
 MATHESIUS, WALTHER, Asst. Supt., Blast Furnaces, South Works, Ill. Steel Co.,
 South Chicago, Ill.
 MERRILL, ERNEST MARTIN, Cons. Engr., Beckley, W. Va.
 MUIR, DOWNIE DAVIDSON, JR., Min. Engr., U. S. Smelt., Ref., & Min.
 Exploration Co., Juneau, Alaska.
 NEILLY, BALMER, Supt. Penn-Canadian Mine, Cobalt, Ont., Canada.
 NOLAN, JOSEPH P., Min. Engr., 308 Lewisohn Bldg., Butte, Mont.
 NOWLAND, RALPH C., Field Engr., D. C. Jackling, 1800 Hobart Bldg.,
 San Francisco, Cal.
 O'NEAL, FRANK E., Engr., Jasper Park Collieries, Ltd., Pocahontas, Alta., Canada.
 ORR, CHESTER A., Supt., Central Furnaces, American Steel & Wire Co.,
 Cleveland, Ohio.
 PUTSCH, AUGUST, Supervising Engr., Lehigh Coke Co., So. Bethlehem, Pa.
 SHARP, JOHN ROY, Genl. Supt. of Mines, Consolidated Indiana Coal Co.,
 Sullivan, Ind.
 SMITH, JAMES H., JR., Engr., Cerro de Pasco Min. Co., Cerro de Pasco, Peru.
 SOMMERVILLE, A. O., Div. Supt., Pennsylvania Coal & Coke Corp., Clearfield, Pa.
 SYLVESTER, FRANK McCLELLAN, Genl. Mgr., Granby M., S. & P. Co.,
 813 Birks Bldg., Vancouver, B. C., Canada.
 TUBBY, CHARLES W., Dist. Mgr., International Steam Pump Co.,
 703 Commerce Bldg., St. Paul, Minn.
 WILLIAMS, WAKELEY AYLESWORTH, Supt. of Smelters, Granby Cons. M., S. & P. Co.,
 Anyox, B. C., Canada.
 WYLY, ARTHUR JAMES, Met. Engr., Lebong Tandai, Benkoelen, Sumatra.

Associate Members

- BACON, HERBERT M., Oil Engineering, 58 Sutter St., San Francisco, Cal.
 VAUCLAIN, ANDREW C., Vice-Pres., Southwark Fdy. & Mch. Co.,
 430 Washington Ave., Philadelphia, Pa.

Junior Members

BONSB, ROY SAMUEL, Student.....	Columbia School of Mines, New York, N. Y.
BURNS, JAY JOSEPH, Student.....	Colorado School of Mines, Golden, Colo.
FOGG, LEWIS WARNER, JR., Student.....	Penn. State College, State College, Pa.
HARBACH, HERBERT MOORE, Student.....	Penn. State College, State College, Pa.
HUTCHISON, ROBERT M., Student.....	Penn. State College, State College, Pa.
TARR, RUSSELL STORY, Student.....	Cornell Univ., Ithaca, N. Y.
WOOD, FLETCHER H., Student.....	Colorado School of Mines, Golden, Colo.

Changes of Status—Associate to Member

APPLEGREN, J. O. W.....	Loops via Marklesville, Alpine Co., Cal.
CROWTHER, J. S., JR.....	Care Toledo Furnace Co., Toledo, Ohio.

Junior to Member

MEANS, ALAN HAY.....	103 St. James Ave., Boston, Mass.
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CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Jan. 10 to Feb. 10, 1915, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

William Seymour Arthur, Williston, N. D.

Proposed by J. C. Roberts, Frank M. Taylor, A. K. McDaniel.

Born 1871, Newark, N. J. 1889, Public and high schools in Idaho Springs, Colo. 1890-91, Univ. of Denver. Course in metal mining, International Correspondence Schools, nearly completed. 1890-91, Platte Valley Irrigation Co., Denver, Colo. 1891-1904, Fairbanks, Morse & Co., Denver, Colo., and mines in Clear Creek County, Colo., as part owner. 1904-15, U. S. Reclamation Service.

Present position: Mgr., North Dakota Pumping Project, Coal Mine and Power Plant.

Hubert Raymond Banks, Humboldt, Ariz.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1891, Toronto, Canada. 1914, grad. in Mining Engrg., University of Toronto; B. Sc. 1909, Miller Lake O'Brien Mine, Gowganda, Ontario. 1915, assaying and surveying; 1915, after graduation Asst. Manager.

Present position: Staff of Consolidated Arizona Smelting Co.

Carl G. Barth, Jr., Philadelphia, Pa.

Proposed by Benjamin L. Miller, H. Eckfeldt, Henry S. Drinker.

Born 1884, Philadelphia, Pa. Public school and preparatory course. 1908, Lehigh University; E. M. 1908, Smith & Furbush Machine Co., during installation of Taylor system of scientific management. Miller Lock Co., Philadelphia, Pa. installing tool room and checking system for tools. 1909, Mucker, drill runner, hoist engr., assayer and surveyor, Cobalt Central Mines, now the Penn Canadian. Drill

runner, surface boss and surveyor, Badger Mines. Machine runner and hoistman during sinking operations, Lumsden Mines, Ltd., Cobalt Ontario. 1910, Draftsman and surveyor, Routly & Summers, Haileyburg, Ontario. Mining claim and town site surveying, mapping, etc., same company, Porcupine, Ont. 1911, Customs assayer and assistant in field on examinations, Porcupine Assay Co. 1911, Assistant to Supt., Standard Gold Mines, Ltd. 1912, Mill construction and operation, Hollinger Gold Mines, Ltd., Concentrator and Cyanide Plants. 1913-14, Genl. Supt., The La Rue Fluorspar Co., Marion, Ky.

Present position: Not employed.

Arthur Francis Bassett, Humboldt, Ariz.

Proposed by G. M. Colvocoresses, J. V. D. Gray, Archie Scott.

Born 1889, Leominster, Mass. 1908-07, New Haven High School. 1907-08, Mercersburg Academy. 1908-11, Sheffield Scientific School, Yale Univ.; Ph. B. 1911-12, Montana State School of Mines; E. M. Two months, mining underground at Butte Superior. About six months, underground in different mines of Anaconda Copper Min. Co. Six weeks, sampling and miscellaneous work during tests of Hancock jig for the Elm Orlu Min. Co. Fifteen months at Butte, Mont., Colusa Parrot Ore Testing Plant.

Present position: 1914 to date, Asst. to Mill Supt., Cons. Ariz. Smelting Co.

Robert M. Betts, Cornucopia, Ore.

Proposed by Benjamin B. Lawrence, J. E. McAllister, Frank Lyman.

Born 1879, Chicago, Ill. 1904, Studied with Gelasio Caetani, Met., assayer, mill work, Atlas Gold Min. Co., Buffalo Hump, Idaho. 1905-06, Bunker-Hill & Sullivan Min. & Concentrating Co., Kellogg, Idaho. Mill work and assaying. Frank C. Lowing, Min. Engr., Toronto, Canada.

Present position: Mgr. of Cornucopia Mines Co.

William Brenton Boggs, Douglaston, N. Y.

Proposed by F. A. Eustis, Augustus H. Eustis, W. E. C. Eustis.

Born 1882, Wilmington, Del. 1889-96, Public schools; 1896-1900, High School, Washington, D. C. 1900-04, Mass. Inst. of Tech.; B. S. 1904-05, Virginia Smelting Co., West Norfolk, Va. 1905-06, Anita Copper Mines Co., So. America, Cocarit, Sonora, Mexico. 1906-07, with Frank Husted, Parral, Chih., Mexico. 1907-08, United States Metals Refining Co., Chrome, N. J.

Present position: 1908 to date, Asst. to Head of Smelting Dept., Nichols Copper Co., Laurel Hill, N. Y.

C. E. Bushnell, Butte, Mont.

Proposed by J. L. Bruce, Frank R. Wicks, W. L. Creden.

Born 1882, Noble, Mich. 1903, Grad. Bronson High School, 12 grades. 1903-06, Three years of a four-year mechanical course, Michigan Agricultural College. 1906, Chemist, Michigan Experiment Station. 1906, Chemist, Michigan Sugar Co. 1907-08, Engr. and draftsman, Ducktown Sulphur, Copper & Iron Co. 1908, Four months, draftsman, Tennessee Copper Co. 1908-10, Prospecting in Utah. 1910, Draftsman, U. S. Smelt., Min. & Ref. Co. 1911, 18 months, drafting for Kelly Filter Press Co. 1911, Draftsman, Utah Copper Co.

Present position: 1912 to date, Draftsman, Butte & Superior Copper Co.

David Amile Cavagnaro, Forest, Sierra Co., Cal.

Proposed by F. F. Thomas, David McClure, O. B. Amsden.

Born 1876, San Andreas, Cal. 1891, Grad. grammar school, in San Andreas. 1891-97, Assay office, surveying and underground mining. 1897-98, special student, Univ. of Cal. 1898-99, Gwin Mine, handling heavy ground. 1899-1903, Grad. Univ. of Cal.; B. S. 1903-04, Assayer, sampler and working in quartz mill, Gwin Mine Co., Cal. 1904-05, Supt., Clover Leaf Mine, S. D. 1905-06, Experimental work, cyaniding and filter pressing, Homestake Mines, S. D. 1906-08, Dredging and goldman, Yuba Cons. Gold Fields, Cal. 1908-09, Goldman, Yukon Gold Co., Y. T. 1909-13, Supt., Channel Mining Co., Cal.

Present position: 1913 to date, Genl. Supt., Wisconsin North Fork Gravel Mines.

A. Newton Cole, Clarion, Pa.

Proposed by H. M. Kanarr, H. J. Hinterleitner, E. W. Hess.

Born 1886, Sharon Center, Pa. 1903-04, Mercersburg Academy, Mercersburg, Pa. 1904-06, Leland Stanford, Jr., Univ., departments of Geology and Mining and Chemistry. 1910-11, Draftsman, Shawmut Mining Co., St. Mary's, Pa. 1911-13, Draftsman and transitman, Alleghany River Mining Co., Kittanning, Pa.

Present position: 1913 to date, General Supt. of Mines, Pennsy Coal Co., Franklin, Pa.

Edward B. Corbet, San Francisco, Cal.

Proposed by D. M. Folsom, H. W. Young, James C. Ray.

Born 1889, Grand Forks, N. D. 1913, Stanford Univ.; A. B. 1912, Universal Oil Co. 1913-14, Geol., for F. W. Kimble. 1914, Placer Mining, Plumas County.

Present position: Private practice, preliminary examination work.

H. A. Coy, Mascot, Tenn.

Proposed by J. N. Houser, J. L. Bruce, H. S. Kimball.

Born 1890, Westboro, Ohio. General education acquired at preparatory school. Technical education acquired by practical experience. 1907-12, Levelman, transitman, topographer, concrete and tunnel inspector, Louisville & Nashville R. R.

Present position: 1912 to date, Mine Supt., American Zinc Co. of Tenn.

John Townsley Knowles Crossfield, Chuquicamata, Chile.

Proposed by Pope Yeatman, Henry Hay, Frederick Hellmann.

Born 1888, Cachar, Assam, India. 1909-13, McGill University; B. S. in Min. Engrg. 1907, Preliminary Survey, Grand Trunk Pacific Ry., Canada. 1908, Everett Silver Cobalt Syndicate. 1909-11, Mond Nickel Co., Sudbury, Canada. 1912-13, Canadian Geological Survey, Ottawa. 1914, Nevada Consolidated Copper Co.

Present position: 1914 to date, Surveyor, Chile Exploration Co.

Donald B. Doyle, New York, N. Y.

Proposed by Chester W. Washburne, S. H. Ball, Lucius W. Mayer.

Born 1885, Mt. Oliver, Pa. 1905, Princeton Univ.; A. B.; 1906, A. M. 1906, Colorado School of Mines; E. M. 1909-11, Engr., Chino Copper Co., Santa Rita, N. M.

Present position: 1912 to date, Engr., explorations in Africa, Société International Forestière et Minière du Congo.

John Lane Dynan, Tonopah, Nev.

Proposed by Horace V. Winchell, H. Eckfeldt, Benjamin L. Miller.

Born 1890, Norwich, N. Y. 1905, Grad. Bethlehem Preparatory School, Bethlehem, Pa. 1909, Lehigh University; E. M. 1909, Asst. Engr., Bessemer Coke Co., Maestown, Pa. 1910, Asst. in Physics, Lehigh University, So. Bethlehem, Pa. 1910-11, Worked underground in Idaho, Utah, and British Columbia. 1911-12, Worked in mill, Tonopah Mining Co., Millers, Nev. 1912, Assayer, Jumbo Extension Mining Co., Bonnie Clare, Nev.

Present position: 1912 to date, Min. Engr., Tonopah Extension Mining Co.

Charles Allen Fellencer, Barranquilla, Colombia.

Proposed by C. J. London, F. Lynwood Garrison, Thomas T. Read.

Born 1891, Bethlehem, Pa. 1913, Lehigh University; E. M. 1913-14, Laboratory work, Miami Copper Co.

Present position: Engr., Colombia Exploration Syndicate, Inc.

Albert Larne Ferris, Metcalf, Ariz.

Proposed by Charron M. Staples, R. B. Earling, Theodore W. Quayle.

Born, 1888, Grand Rapids, Mich. 1908, Grad., Petoskey High School, Mich. 1911, Michigan College of Mines; B. S. and E. M. 1911, Started work sampler and draftsman, transitman and shift boss, Shannon Copper Co.

Present position: Chief Engr., Metcalf Division, Ariz. Copper Co.

Frederick B. Forbes, Granada, Nicaragua.

Proposed by J. C. Branner, H. W. Young, G. H. Clevenger.

Born 1889, Portage, Wis. 1909-11, Univ. of New Mexico. 1911-13, Stanford Univ., Technical. 1913, Nevada Cons. Copper Co. 1914, F. A. Pellas & Co., Granada, Nicaragua.

Present position: Asst. Supt., Escandalo Mine.

M. Shozo Hachita, Wilkes-Barre, Pa.

Proposed by J. M. Humphrey, Paul Sterling, Charles Ensian.

Born 1869, Japan. 1894-98, Mt. Hermon School, Mt. Hermon, Mass. 1898-1902, Lehigh University, So. Bethlehem, Pa. 1902-03, Chemist and Min. Engr., Mexican Gulf Coal & Transportation Co., Indian Ter. 1903-07, Transitman; 1907-10, Asst. to Mining Engr.; 1910-13, Chemist, Lehigh Valley Coal Co., Wilkes-Barre, Pa.

Present position: 1913 to date, Chemist and Editor, *Employee's Magazine*, Lehigh Valley Coal Co.

Charles James Hall, London, E. C., England.

Proposed by F. C. Newton, L. D. Ricketts, P. A. Mosman.

Born 1877, Adelaide, So. Aust. 1882-93, Private tuition. 1894-97, Adelaide Univ., So. Aust. 1897-1905, Chem., Mt. Lyell Min. & Ry. Co. 1905-06, Chemist, Tacoma Smelt. Co. 1906-07, Asst. Supt., San Bruno Plant, Selby Smelt. & Lead Co. 1907-08, Chemist, Everett Plant, Am. S. S. Co. 1908-10, Supt., Balaklala Cons. Copper Co. 1911, Foreman, Suriana Min. & Smelt. Co. 1911-12, Met., Reforma Min. & Mill. Co. 1912-13, Met., Swansea Cons. Gold & Copper Min. Co.

Present position: 1913 to date, Met., Caucasus Copper Co.

Mortimer Louis Hall, Latouche, Alaska.

Proposed by J. C. Branner, H. W. Young, D. M. Folsom.

Born 1889, Bucyrus, Ohio. 1907-10, Student, Chemistry and Engrg., Stanford Univ. 1911, Student of Mining, Colorado College. 1911-13, Stanford Univ.; A. B. in mining and metallurgy. 1910-11, Bench Chem., American Beet Sugar Co., Chino, Cal.

Present position: 1913 to date, Surveyor, Beatson Copper Co.

Arthur Ludwick Halvorsen, Perth Amboy, N. J.

Proposed by H. Foersterling, F. C. Newton, Frederick Jaeger.

Born 1888, Sunde, Sandefjord, Norway. 1904, Public High School of Sandefjord Technological School of Cristiania, Chemist. 1909, Dr. Emil Collett's Laboratory, Christiania, Norway. 1910, Legacion de Noruega, Habana, Cuba.

Present position: Chemist, Roessler & Hasslacher Chemical Co.

Perry G. Harrison, National, Nev.

Proposed by H. L. Hollis, D. P. Hynes, H. G. S. Anderson.

Born 1885, Minneapolis, Minn. 1904-05, Mich. College of Mines. 1905-06, Prospector, miner, laborer, bookkeeper, shifter, gen. surf., Kerr Lake Min. Co., White Hargrave Co., Harris Mine, Cobalt, Ont. 1906-07, Mich. College of Mines. 1907-08, Min. Engr., Meriden Iron Co., Hibbing, Minn. 1908-09, Mich. College of Mines; E. M. 1909, Millhand, miner, Calumet & Hecla, Calumet, Mich. 1909-10, Post-graduate, Columbia College. 1910-11, Min. Engr., Minn. Meriden Iron Co., Hibbing, Minn.

Present position: 1911 to date, Supt., National Mines Co.

John E. Hogan, Grass Valley, Cal.

Proposed by William Hague, Robert H. Bedford, Arthur DeW. Foote.

Born 1867, Grass Valley, Cal. 1884, High School, Grass Valley. 1888-1893, Min. and mill work, Grass Valley. 1893, Millwork, Cripple Creek. 1894-97, Miner, Grass Valley, Cal. 1897-1907, Shift boss, North Star Mines Co. 1907, Mine foreman, Central America.

Present position: 1908 to date, Mine foreman, North Star Mines Co.

Wu Kwong, Peking, China.

Proposed by Chieh Ho, Philip Jay Lonergan, Jr., George I. Adams.

Born 1890, Fukien Province, China. 1903-09, Grad. Shantung Provincial College; B. Sc. 1910-13, Peking Government University, Mining and metallurgical department; M. E. 1913, practical student in the Kailan Colliery, Chili, North China. 1913-14, Mgr. and Engr., Hsing-Chang Coal Min. Co., Western Hill, via Peking. 1912, Member of the Chinese Institute of Civil Engineers.

Present position: Secretary, Chinese Institute of Mining Engineers.

F. Larogue.

Proposed by W. L. Honnold, Palmer Carter, C. E. Knecht.

Born 1889, Mauritius. 1906, Cachelier Ire partie Sciences langues vivantes Bordeaux, France, while attending the Lyceum. 1907, Scientific degree Polytechnic Preparatory School, Brooklyn. 1911, Columbia Univ.; E. M.

Present position: 1912 to date, Shift boss, Durban Roodepoort Deep Gold Mining Co., Ltd., Transvaal.

Walter Stuart Larsh, Chuquicamata, Chile.

Proposed by Edwin S. Berry, Robert Marsh, Jr., E. A. C. Smith.

Born 1883, Cheyenne, Wyo. 1889-97, Emerson School, Denver, Colo. 1897-1900, Manual Training High School, Denver, Colo. 1900-04, Colorado School of Mines; E. M. 1904-05, Stratton's Independence, Ltd., Victor, Colo. 1906-08, Cumberland Ely Copper Co., Ely, Nev. 1909-10, Scouting, Nevada. 1911, Los Angeles Aqueduct, Los Angeles, Cal. 1911-13, Braden Copper Co., Rancagua, Chile.

Present position: M. Guggenheim's Sons.

Arthur Solomon LeFevre, Johannesburg, Transvaal, S. Africa. *

Proposed by W. L. Honnold, Palmer Carter, C. E. Knecht.

Born 1887, London, England. 1903, Matriculated, Univ. Cape of Good Hope. 1904, First mining examination. 1905, Second mining examination. 1906, Third mining examination. 1907, Diploma and associateship, South African School of Mines and Technology (A. S. A. S. M. T.). 1911, Certificate of competency as mine surveyor (S. A. Mines Dept.). 1908-12, Asst. and chief sampler, also asst. surveyor, Knight Central, Ltd. 1912, Asst. surveyor, New Kleinfontaine Co., Ltd. 1913, Chief surveyor, Randklip, Ltd.

Present position: 1913 to date, Asst. surveyor, New Klenfontein Co., Ltd.; author of "Mine Sampling and Ore Valuation," published by "Transvaal Leader," August, 1913.

Frederick F. Lines, Sparrows Point, Md.

Proposed by Edward S. Cook, Quincy Bent, F. W. Wood.

Born 1880, Great Bend, Pa. 1898-1902, Grad., Lehigh Univ.; M. E. 1902-04, Learner in Bessemer Dept. 1904-06, Engr. of Tests. 1906-12, Supt., Bessemer Dept., Maryland Steel Co.

Present position: 1912 to date, Supt., Bessemer and Open Hearth, Maryland Steel Co.

James C. McNabb, Humboldt, Ariz.

Proposed by G. M. Colvocoresses, D. J. Evans, Archie Scott.

Born 1886, Steubenville, Ohio. 1892-1900, St. Mary's Parochial School, Tiffin, Ohio. 1900-02, Public school. 1902-06, High school, Fostoria, Ohio. 1907-11, Ohio State Univ., School of Mines, Columbus, Ohio; E. M. 1911, Assayer, Candor Mines Co., Candor, N. C. 1911-13, Chemist, United States Metals Ref. Co., Chrome, N. J.

Present position: 1914 to date, Assayer and Chemist, Consolidated Arizona Smelting Co.

Dr. John Alexander Mathews, Syracuse, N. Y.

Proposed by Henry M. Howe, Bradley Stoughton, Charles F. Rand, Albert Sauveur, A. A. Stevenson.

Born 1872, Washington, Pa. 1893-1902, Washington and Jefferson; B. Sc., D. Sc. 1905-08, Columbia Univ.; M. A., Ph. D. 1900-01, research student, Royal School of Mines, London. 1887-88, Sometime University Fellow in Chemistry, Columbia; 1900-02, and Barnard Fellow. 1900-01, Carnegie Research Scholar, Iron and Steel Inst. of Gt. Britain, and first Carnegie Gold Medallist of I. & S. Institute. 1903, Met., Crucible Steel Co. of America. 1904-08, Asst. Mgr., Sanderson Bros. works, Crucible Steel Co. of America.

Present position: 1908 to date, Genl. Mgr., Halcomb Steel Co.

Harry L. Mead, New York, N. Y.

Proposed by C. C. Burger, B. B. Thayer, Thomas T. Read.

Born 1880, Sycamore, Ill. 1900-01, Grad., Armour Scientific Acad. 1901-03, Armour Inst. of Tech., courses of Civil and Mech. Engr. 1903-05, Columbia Univ., School of Mines, advanced standing; M. E. 1905-06, Asst. Prof. Min. and Geol., State Univ. of Washington, Seattle. 1906, Asst. Supt., Pillsbury, Bennett & Long-

year interests, Hibbing, Minn. 1906-08, Engr. and Supt., Avino Mines of Mexico, Avino, Dgo., Mexico. 1908, Mgr., New York Co., Santa Eulalia, Chih., Mexico. 1909, Examinations in Mexico. 1909-12, Asst. Mgr., Lefrange Mines Co., Weaver-ville, Cal.

Present position: Examining Engr. on placer deposits for Lewisohn Bros., Breitung & Co., John Hays Hammond, etc.

Carl Weaver Mitman, Washington, D. C.

Proposed by Benjamin L. Miller, H. Eckfeldt, Henry S. Drinker.

Born 1889, Newbury, Pa. 1909, Lehigh Univ., B. A.; 1911, E. M. 1912, one year, Post Graduate, Princeton Univ. 1911-12, Asst. in Geology, Lehigh Univ. 1912-13, Asst. in Mineralogy, Princeton Univ. 1913-14, Foreman, Oxide Dept., New Jersey Zinc Co., Palmerton, Pa.

Present position: 1914 to date, Aid in Mineral Technology. U. S. National Museum.

William Llewelyn Morgan, E. St. Louis, Ill.

Proposed by R. Y. Williams, E. A. Holbrook, H. H. Stoek.

Born 1874, South Wales. General education, public schools, South Wales, until 1888. 1891-1903, Technical education in coal mining, geology, applied mechanics, and mathematics, South Wales. Technical education was acquired under the Glamorgan County Council educational department (technical). This department provided for the education of working men by continuation schools, held at night. These schools were provided with grants from the government under the South Kensington Department, London. 1890-1911, worked in every position in the mine, except as driver, and being in active charge of a mine. 1911-14, State Inspector, Eighth District, State of Illinois.

Present position: 1914 to date, Instructor, Illinois Miners' and Mechanics' Institute, University of Illinois.

John Thomas Morris, Lowe or Springton, W. Va.

Proposed by H. Eckfeldt, Benjamin L. Miller, Henry S. Drinker.

Born 1887, Wilkes-Barre. 1902, Hanover Township High School, Luzerne Co., Pa. 1913, Lehigh Univ., special student. 1907-08, Cons. Engr., Quinnimount, W. Va. 1909-11, Engineering Dept., Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1911-12, Asst. Chief Engineer Coal Mines Tenn. Coal, Iron & Railroad Co.

Present position: Genl. Supt., Weyanoke Coal & Coke Co. and S. J. Patterson Pocahontas Co.

Emil Mosonyi, San Salvador, C. A.

Proposed by Anthony F. Lucas, M. R. Campbell, Alfred H. Brooks.

Born 1876, Szombathely, Hungary. 1896, grad., in Szeged, Hungary; B. S. 1896-99, Univ. of Technic at Budapest. 1899-1900, served as Voluntario at the Austro-Hungarian Navy and became non-commissioned officer. 1900-01, Returned to the Univ. of Engineering. 1901-05, Traveled the larger part in the U. S., holding several positions and making studies always in engineering lines in different lines. Represented W. A. Miller Mfg. Co. at the St. Louis World's Fair. At the end of the year with the St. Louis Smelting & Refining Co. at Desloge, Mo.; redesigned the mill work, etc. 1905-14, traveled exclusively in Central and South America. for N. Y. Min. Exploration Co. and Tropical Min. & Explor. Co. Member of the Geographical Societies N. Y. and Washington, D. C.

Present position: Technical Director, Salvador Deep Well Boring Co.

M. H. Newman, Mascot, Tenn.

Proposed by J. N. Houser, J. L. Bruce, H. S. Kimball.

Born 1878, Trempealeau, Wis. 1897-1901, Univ. of Wisconsin; A. B. 1902-03, Univ. of Wisconsin, graduate work. 1902-03, Oliver Iron Mining Co. 1904-07, Vinegar Hill Zinc Mining Co. 1907-12, Oliver Iron Mining Co.

Present position: 1912 to date, American Zinc, Lead & Smelting Co.

Archie J. Orem, Ludwig, Nev.

Proposed by George A. Packard, Charles E. Locke, H. O. Hofman.

Born 1881, Lathrop, Mo. 1907-10, Mass. Inst. of Tech., special student, Min. Engrg. course. 1897-99, New Red Mining Co., Bingham Canyon, Utah. 1900-05, Supt., Utah Apex Mining Co., Bingham, Utah. 1905-06, Supt., Idaho Development Co., Silver City, Idaho. 1906-07, Supt., North Utah Mining Co., Bingham, Utah. 1911-12, Asst. Mgr., Castle Valley Coal Co., Mohrland, Utah.

Present position: 1912 to date, Genl. Supt., Nevada Douglas Copper Co.

Ernest Jerome Rossbach, St. Louis, Mo.

Proposed by S. S. Clarke, C. J. Adami, A. E. Ring.

Born 1890, Milwaukee, Wis. 1897-1904, Grade Schools, Milwaukee; Boone, Iowa; St. Paul, Minn.; Dubuque, Iowa. 1904-08, High School, Dubuque, Iowa, and Hyde Park, Chicago. 1908-12, Univ. of Ill.; B. S. in Min. Engrg. 1910, Engr., Illinois Tunnel Co., Chicago, Ill. 1911, Sales Dept., Marquette Portland Cement Mfg. Co., Chicago.

Present position: Sales Engr., Sullivan Machinery Co.

William Henry Seagrave, Kennecott, Alaska.

Proposed by H. Foster Bain, Stephen Birch, E. A. Cappelen Smith.

Born 1875, Susanville, Cal. 1892, Grad., High School, Reno, Nev. 1896, Grad., School of Mines, Univ. of Nevada; B. S. 1896-97, Employed in copper smelter, Mountain Copper Co., Ltd., Keswick, Cal. 1897-99, Supt., Mammoth-Garfield Gold Min. Co., Shasta County, Cal., and Alta Sierra Gold Min. Co., Sierra County, Cal. 1900-02, Mine Mgr., United Rhodesia Gold Fields, Ltd., Rhodesia. 1907, Underground Mgr., Simmer & Jack Prop. Mines. Transvaal, So. Africa. 1907-10, Genl. Supt., Cumberland Ely Copper Co., Ely, Nev.

Present position: Mgr., Kennecott Mines Co.

Harold Morgan Smyth, Pottsville, Pa.

Proposed by G. R. Wood, Frank A. Hill, James Archibald, Jr.

Born 1891, Wilkes-Barre, Pa. Pottsville public schools. 1907-08, Bethlehem Prep. School, Bethlehem, Pa. 1908-12, Lehigh University, So. Bethlehem, Pa.; E. M. 1912, Philadelphia & Reading Coal & Iron Co., Eng. Dept., Pottsville, Pa.

Present position: 1913 to date, Asst. Supt., St. Clair Coal Co.

Paul Robert Snyder, Bethlehem, Pa.

Proposed by Henry S. Drinker, H. Eckfeldt, Benjamin L. Miller.

Born 1888, Bethlehem, Pa. 1907, Grad., Bethlehem High School. 1911, Grad., Lehigh Univ.; M. E. in Metallurgical Engrg. 1911-13, International Nickel Co., Copper Cliff, Canada. 1913-14, Union Carbide Co., Sault Ste. Marie, Mich.

Present position: 1914 to date, Foreman, Heat Treatment Dept., Bethlehem Steel Co.

Frank Lawrence Stack, Chuquicamata, Chile.

Proposed by Pope Yeatman, Henry Hay, Frederick Hellmann.

Born 1887, San Diego, Cal. Univ. of California; B. S. Engrg. Lord & Young, Honolulu, Hawaii. Dept. of Public Works, Honolulu. Snow Mt. Water & Power Co., Lake County, Cal. Belcher Mine, Republic, Wash. Eagle Shawnut Mine, Toulumne County, Cal. Nevada Cons. Copper Co., McGill, Nev.

Present position: Chemist and Assayer, Chile Exploration Co.

Charles H. Taylor, Norman, Okla.

Proposed by Thomas T. Read, C. K. Leith, W. H. Emmons.

Born 1877, Dwight, Kan. 1905, B. S.; 1909, M. S., University of Chicago. 1907-08, Prof. of Science, State Normal School, River Falls, Wis. 1909-10, Associate Prof. of Geology, Univ. of Okla. 1910-11, Director, School of Mining Geology and Prof. of Mineralogy, Univ. of Okla. 1911-12, Director, School Mining Geol. and Head of Dept. of Geol., Univ. of Okla. 1909-15, Junior Geologist, U. S. G. S., and Consulting Oil Geologist.

Present position: Director, School Mining Geology, Prof. of Geology, Univ. of Oklahoma.

Arthur Clark Terrill, Albany, N. Y.

Proposed by David H. Newland, J. F. Kemp, Charles P. Berkey.

Born 1882, Woodbury, Conn. 1901, Grad., Colorado Springs High School. 1905, Colo. School of Mines; E. M. 1914, Columbia Univ.; A. M. 1899, H. V. Wadell, Broker, Colorado Springs, Colo. 1902-03, Portland Chlorination Plant, Colo. Springs, Colo. 1905, E. C. Woodward, Assayer, Colo. Springs. 1905, Slimes and Leaching Dept., Dorcas Cyanide Plant, Florence, Colo. 1905-06, Supt., Doctor-Sack Pot Mine, Cripple Creek, Colo. 1906-08, Prof. of Min., Met., and Geol., Univ. of Oregon. 1909-11, Gen. Secy., Y. M. C. A., Eugene, Ore. 1911-12 Asst. Ed'l. Secy., Y. M. C. A., Los Angeles, Cal. 1912-13, Principal Min. and Surveying Dept., Y. M. C. A. Technical school, Los Angeles, Cal. 1913-14, Grad. student in Geol. Min. and Met., Columbia Univ., New York.

Present position: Field Representative, N. Y. State Min. Exhibit, Panama Pacific Exposition.

Max Ogilvie Tillard, Johannesburg, Transvaal, S. Africa.

Proposed by W. L. Honnold, Palmer Carter, C. E. Knecht.

Born 1880, South Africa. 1895-98, S. A. College, Cape Town. 1898-1902, S. A. School of Mines, Kimberley and Johannesburg. Diploma of Mining Engineer, Cape. 1902-05, Surveyor, Meyer & Chailton & New York H. M. 1906, Mine captain, Jumpers Deep. 1909, Asst. Mgr., Geldenheim Deep. 1912-13, Mgr., City Deep.

Present position: 1914 to date, Group surveyor, Central Mining & Inv. Corp.

Edward A. Van Horn, Shaft, Schuylkill County, Pa.

Proposed by R. V. Norris, W. J. Richards, Robert A. Quin.

Born 1877, Wilkes-Barre, Pa. 1893, Wilkes-Barre High School. 1898-1901, Surveyor, Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1902-06, Asst. Engr., Susquehanna Coal Co., Nanticoke, Pa. 1906-07, Engr., Lytle Coal Co., Minersville, Pa. 1908-11, Engr., Summit Branch Mining Co., Lykens, Pa.

Present position: 1911 to date, Supt., Susquehanna Coal Co.

Christian M. Vrang, Stanford Univ., Cal.

Proposed by James C. Ray, C. F. Tolman, Jr., G. H. Clevenger.

Born 1887, Ross, Cal., Student at Stanford Univ. 1910, Engineering staff, Ray Cons. Copper Co. 1911, Engineering staff, Guggenheim Exploration Co., Alaska. 1912, Examination of Placers, Ruby, Alaska.

Present position: Student, Stanford University.

Charles Newton Whitaker, Jr., Monrovia, Cal.

Proposed by Theodore H. M. Crampton, W. G. Haldane, Harry J. Wolf.

Born 1889, Marion, Kan. 1914, Colorado School of Mines; E. M. 1913, sampler and crusherman, Cornucopia Mines Co.; "Roustabout" in cyanide mill, Cornucopia, Ore. 1914, Asst. assayer and Kelly pressman, Butte & Duluth Mining Co., Butte, Mont. 1914, Asst. Engr., Arizona Copper Co., Metcalf, Ariz.

Present position: Not employed.

Rollin Henry Wilbur, Philadelphia, Pa.

Proposed by Henry S. Drinker, H. Eckfeldt, Benjamin L. Miller.

Born 1863, Bethlehem, Pa. 1880-84, Lehigh University. 1907-date, Vice-Pres., Lehigh Coal & Navigation Co.; Alliance Coal Mining Co. Vice-Pres. and Genl. Mgr., Lehigh & New England R. R. 1911 to date; Vice-Pres., Lehigh Navigation Electric Co. Member of Institute from 1886 to 1898.

Present position: 1913 to date, Vice-President, Harvard Electric Co.

Howard I. Young, Carthage, Mo.

Proposed by J. H. Polhemus, J. N. Houser, A. K. McDaniel.

Born 1889, Jasper County, Mo. 1904, Grade and High School studies. 1905, Business College. 1903-04, summers, Mt. Vernon Min. Co., Chicago, Ill. 1906-08, Bookkeeper, Carthage F. & M. Works, Carthage, Mo. 1908-14, Cashier and Accountant, American Zinc, Lead & Smelting Co., Cartersville, Mo.

Present position: Mgr., Missouri Mines, American Zinc, Lead & Smelting Co.

Associate Member

Huan Ting Liang, New York, N. Y.

Proposed by Thomas T. Read, Theodore Sternfeld, L. V. Emanuel.

Born 1892, Chang Sha, China. 1909-11, Hunan Polytechnic Institute. 1913-14, Royal School of Mines, London. 1911-13, Chemist, Hunan Bureau of Mines.

Present position: Agent of Wah Chang Co. in New York City.

Junior Members

Lawrence A. Callaway, Cambridge, Mass.

Proposed by Edward D. Peters, Charles H. White, Albert Sauveur.

Born 1885, Marshfield, Mo. 1910, Univ. of Ariz.; S. B. 1912, Missouri School of Mines. 1913, Kentucky State Univ. 1914, Harvard University; M. E.

Present position: Student, Harvard Mining School.

Clark Bailey Carpenter, Lawrence, Kan.

Proposed by E. S. Dickinson, A. W. Smith, C. M. Young.

Born 1888, Girard, Kan. 1907-08, Grad., Girard High School. 1910, Tracklayer, Timberman and night boss at Primero, Colo., Colorado Fuel & Iron Co. 1914, Girard-Cherokee Coal Co., Mindenmines, Mo.

Present position: Student, Univ. of Kansas.

James P. Clarke, Stanford Univ., Cal.

Proposed by J. C. Branner, D. M. Folsom, C. F. Tolman, Jr.

Born 1890, Helena, Ark.

Present position: Student, Leland Stanford Junior Univ.

Benjamin Clark Essig, Golden, Colo.

Proposed by F. W. Traphagen, Harry J. Wolf, William R. Chedsey.

Born 1892, Elsie, Mich. 1909-10, Michigan Agricultural College. 1910-11, 1912-15, Colo. School of Mines. 1912, Surveying and sampling, Liberty Bell Gold Mining Co. 1914, Trammig and mucking, Portland Gold Mining Co.

Present position: Student, Colo. School of Mines.

Ernst Hertel Ruebel, Rolla, Mo.

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1891, St. Louis, Mo. 1906-08, Manual Training School, St. Louis. 1909-10, Soldan High School, St. Louis. 1912, Granby Min. & Smelt. Co., Construction work, E. St. Louis. 1913, National Zinc Co., Kansas City, Chemist.

Present position: Student, Missouri School of Mines.

Hsiang Tsai, Golden, Colo.

Proposed by F. W. Traphagen, Harry J. Wolf, William R. Chedsey.

Born 1891, Hanhow, China.

Present position: Student, Colorado School of Mines.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Jan. 10 to Feb. 10, 1915. This list, together with the list published in *Bulletin* Nos. 88 to 98, April, 1914, to Feb., 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of Feb. 10, 1915.

ABADIE, AMILE R.	Box 291, Porterville, Tulare Co., Cal.
AMBROSIUS, J. R.	Apartado 1343, Mexico City, Mexico.
AUSTIN, E. A.	Flat City, Itditarod District, Alaska.
BAGG, RUFUS M.	466 Alton St., Appleton, Wis.
BARDWELL, E. S.	960 Ventura Street, Berkeley, Cal.
BARNETT, WILLIAM J.	8 Waterloo Place, Pall Mall, London, S. W., England.
BATCHELLER, STILLMAN	Instructed to hold all mail.
BELL, DONALD A. S.	136 McLaren St., Ottawa, Canada.
BELL, J. M.	Almonte, Ont., Canada.
BERG, H. A.	102 Oakview Avenue, Edgewood Park, Swissvale, Pa.
BETTS, ANSON G.	Box 792, Asheville, N. C.
BLACKMAR, FRANK H.	Care Olimpo del Valle, Mangangué, Colombia.
BLAKE, D. E.	Elks Club, Phoenix, Ariz.
BLAKESLEE, FRANK A.	Care Ashanti Gold Fields, Ltd., Obuasi, Gold Coast Colony, West Africa.
BOLLES, MYRICK N.	2129 Garland Ave., Muskogee, Okla.
BORCHERT, WALTER O.	Austinville, Va.
BORIE, A. E., Chairman Board of Directors, American Road Machinery Co.,	50 Church Street, New York, N. Y.
BOYD, A. W.	1807 6th St., Spokane, Wash.
BOYER, SAMUEL L.	2808 W. Boone Ave., Spokane, Wash.
BRATTON, COREY C., Min. Engr. and Genl. Mgr.,	Shovel Creek Gold Dredging Co., San Francisco, Cal.
BRITT, R. H.	Box 55, Rockland, Maine.

- BRODIE, WALTER M., Min. Engr. and Met., Batopilas Co.,
50 Broad St., New York, N. Y.
- BROWN, D. C. 82 Beaver St., New York, N. Y.
- BRUNTON, FREDERICK K. 138 Prospect Ave., Cripple Creek, Colo.
- BRYANT, ALBERT D., Asst. Chief Oxide East Dept., New Jersey Zinc Co.,
Palmerton, Pa.
- BULL, R. A. Mgr. of Production, Commonwealth Steel Co., Granite City, Ill.
- BURGREN, A. W. 900 61st St., Oakland, Cal.
- CARNAHAN, G. H. Tesitlan Copper Co., 82 Beaver St., New York, N. Y.
- CARSON, JOSEPH A., Allis-Chalmers Manufacturing Co.,
732 Salisbury House, London Wall, London, E. C., England.
- CASE, BENJAMIN H. Boyd Smith Mines, Inc., Mineral, Va.
- CASTLETON, W. A. Mill Supt., Care Utah Apex Mining Co., Bingham, Utah.
- CAZIN, FRANK. 2150 Lafayette St., Denver, Colo.
- CHARLES, W. W., Tonopah Mining Co. of Nevada, 572 Bullitt Bldg., Philadelphia, Pa.
- CHASE, ROGER E. 3005 N. Proctor St., Tacoma, Wash.
- CHIBAS, EDUARDO J., Pres., Guantanamo Electric Co. and Guantanamo Ice Co.,
Santiago de Cuba, Cuba.
- CLARK, E. B., Vice-Pres. and Genl. Mgr., American Sintering Co.,
631 Railway Exchange Bldg., Chicago, Ill.
- COMSTOCK, CHARLES W. 1235 First National Bank Bldg., Denver, Colo.
- COOK, FREDERICK S. 1001 Northwestern Bank Bldg., Portland, Ore.
- CORSE, W. M., Mgr., Bronze Dept., Titanium Alloy Mfg. Co., Niagara Falls, N. Y.
- COSTE, EUGENE. 215 6th Ave. W., Calgary, Alta., Canada.
- COXE, EDWARD H. Apartment 12, The Marion, Knoxville, Tenn.
- COYLE, JOHN A. 1318 Vance Ave., Coraopolis, Pa.
- CREMER, FELIX. Needles, Cal.
- CUNNIFF, BERNARD. 53 State St., Boston, Mass.
- CUNNINGHAM, NOEL. 357 Market St., Bethlehem, Pa.
- DAVELER, ERLE V. Supt. of Mills, Alaska Gold Mines Co., Thane, Alaska.
- DESHAYES, ERNEST V. Baldy, N. M.
- DOAK, S. E. 1502 N. 15th St., Philadelphia, Pa.
- DORR, JOHN VAN N. Met., 30 Church St., New York, N. Y.
- DRESSER, CLARENCE G. Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
- DUNSHIE, B. H. Asst. Genl. Mgr., Anaconda Copper Min. Co., Butte Mont.
- DUNSTER, C. B., Manager of Mines Dept., Breitung & Co., Ltd.,
11 Pine St., New York, N. Y.
- EDWARDS, J. R. Navy Yard, Charleston, S. C.
- ELLIOTT, ROY H. Min. Engr., 437 Holbrook Bldg., San Francisco, Cal.
- EPPLEY, MARION. 39 E. 57th St., New York, N. Y.
- EVELAND, A. J. Care Cia. de Santa Gertrudes, S. A., Pachuca, Hid., Mexico.
- FAHRENWALD, F. A. University of Michigan, Ann Arbor, Mich.
- FARNHAM, LLOYD L. Silver City, S. D.
- FARRANT, JAMES C. Instructed to hold all mail.
- FINK, WILLIAM N., Care Cusi Mexicana Mining Co., Cusiuhiriachic, Chih., Mexico.
- FISHER, HOWELL T., Tunnel Engr., MacKenzie, Mann & Co., Ltd.,
Canadian Northern R. R., Montreal, Canada.
- FORD, HAROLD P. Box 281, Lake Linden, Mich.
- FOUST, THOMAS B. Fce. Supt., The Alabama Co., Ironaton, Ala.
- FOVARGUE, FRANK H. Terlingua, Texas.
- FREYN, H. J. 348 People's Gas Building, Chicago, Ill.
- GAMM, H. F. E. 154 Carmita Ave., Rutherford, N. J.
- GBB, ALLAN. Bel-Air, Alderley Edge, Cheshire, England.
- GIFFORD, STANLEY D. 512 Fifth Avenue, New York, N. Y.
- GITTINS, ARTHUR W. Lynwood Apts., Forbes & Murray Sts., Pittsburgh, Pa.
- GRIPARI, M. G. DE. Baranovka, Gouvernement de Vohlynie, Russia.
- GROSBURG, ALEXANDER. Care Simon I. Patino Tin Mines, Uncia, Bolivia.
- HALL, E. B. 1438 Kellman Ave., Los Angeles, Cal.
- HAMANN, WILLIAM A., JR. 379 Temple Street, New Haven, Conn.
- HAMBLETON, JAMES W., Miner, Cia. Metalurgica de Torreón, S. A.,
Torreón, Coah., Mexico.
- HARDER, EDMUND C. 206 Science Hall, Madison, Wis.
- HAYNES, JUSTIN H. Met. Engr., 319 Boston Bldg., Denver, Colo.
- HEGDEM, ALF G. Petroleum Engr., U. S. Bureau of Mines, Tulsa, Okla.
- HENDERSON, J. McC. Care Campbell, Fairmount Blvd., Eugene, Ore.

- HICKMAN, E. C. P. O. Box 43, East Helena, Mont.
 HIESTER, ARTHUR J. . Min. Engr., Empire Zinc Co., 703 Symes Bldg., Denver, Colo.
 HOLMES, FREDERICK CHARLES. P. O. Box 240, San Juan, Porto Rico.
 HOLTER, EDWIN O. 60 Broadway, New York, N. Y.
 HOWARD, JOHN J. 389 W. Blackwell St., Dover, N. J.
 HUNTLEY, LOUIS G., Geol. and Engr., Johnson & Huntley,
 Grant Blvd. and O'Hara St., Pittsburgh, Pa.
 HUTCHINSON, W. SPENCER. . 1021 State Mutual Bldg., 50 Congress St., Boston, Mass.
 HURD, RUKARD., Minnesota State Tax Commission, State Capitol, St. Paul, Minn.
 HUSTON, HARRY LEE. 634 Mills Bldg., San Francisco, Cal.
 IMHOFF, ALEXANDER. Box 104, Murphy, Cal.
 JACKLING, D. C. Hobart Building, San Francisco, Cal.
 JACKSON, G. T. Thane, Alaska.
 JAMES, C. H. 410 Maskey Bldg., San Francisco, Cal.
 JAMISON, C. E. Cody, Wyo.
 JOHNSON, PERCIVAL. Genl. Mgr., Pulaski Iron Co., Pulaski, Va.
 KANO, SHINICHI, Mgr., Abeshiro Mine, Kawauchi-mura, Shimokita-gun,
 Aomori-ken, Japan.
 KEADY, L. Y. 216-217 Lumbermens Bldg., Portland, Ore.
 KELLEY, ARTHUR L. Instructed to hold all mail.
 KELLEY, C. F. 616 Hennessy Bldg., Butte, Mont.
 KENNEDY, E. P. 63 7th Ave., San Francisco, Cal.
 KENT, WILLIAM. 64 Orange Road, Montclair, N. J.
 KISHMAN, MAURICE W. 101 E. Masonic Ave., Cripple Creek, Colo.
 KLEINSCHMIDT, F. H., Lease on Blue Jacket and Queen Mines,
 Landore, Adams Co., Idaho.
 KNOX, ROGER C. Black Oak Min. & Mill. Co., Soulsbyville, Cal.
 KOEHLER, CARL F. 1228 E. 113th St., Cleveland, Ohio.
 KUELL, C. R. Care Washoe Reduction Works, Anaconda, Mont.
 LADD, DAVID H. Hancock, Mich.
 LANDFIELD, J. B. Bohemian Club, San Francisco, Cal.
 LANDON, STEPHEN L. Box 393, Bisbee, Ariz.
 LASSITER, ROBERT G. First National Bank Bldg., Oxford, N. C.
 LAVERY, V. M. Guasipati, Venezuela.
 LEARY, DANIEL J. 37 E. 49th St., New York, N. Y.
 LEAVELL, JOHN H. 804 Newhouse Bldg., Salt Lake City, Utah.
 LESLIE, HUGH M., Care Affleck & Co., P. O. Box 3906,
 Johannesburg, Transvaal, S. Africa.
 LINDSAY, WILLIAM R. Thane, Alaska.
 LUCKE, P. K., Genl. Supt. of Mines, Care Cia. Minera de Penoles,
 Mapimi, Dgo., Mexico.
 MCCORMICK, V. C. Bergner, Bldg., Harrisburg, Pa.
 MCINTOSH, F. K. 812 E. 8th St., Traverse City, Mich.
 MCLELLAN, J. J. Webb City, Mo.
 MCINTOCK, A. Hermon, St. Lawrence Co., N. Y.
 MANNEHEIM, F. A. L. 76 Warren St., New York, N. Y.
 MARSHALL, GEORGE B. Care Mina La Union, Miramar, Costa Rica.
 MARSHALL, N. C. 611 Brooklyn Ave., Kansas City, Mo.
 MARTIN, S. G., American Min. & Power Co., Martin Min. & Power Co.,
 321 Boston Bldg., Denver, Colo.
 MASON, J. GORDON. Ingenio Rio Canto, Oriente, Cuba.
 MAYNARD, T. POOLE. 1622-D-Hurt Bldg., Atlanta, Ga.
 MEANS, ALAN H. 103 St. James Ave., Boston, Mass.
 MENTZEL, CHARLES. 1935 79th St., Brooklyn, N. Y.
 MERCER, JOHN W. 15 Broad St., New York, N. Y.
 MERRY, F. CHARLES. Kaslo, B. C., Canada.
 MILLWARD, WILLIAM. Suite 7, 1877 E. 97th St., Cleveland, Ohio.
 MITCHELL, D. P. 7 Gracechurch St., London, E. C., England.
 MOORE, L. D. Engr., American Zinc & Chemical Co., Langeloth, Pa.
 NASH, WILLARD H. 297 Delaware Ave., Buffalo, N. Y.
 NEWBERRY, ANDREW W. Min. Engr., 321 Story Bldg., Los Angeles, Cal.
 NOTMAN, ARTHUR. Bisbee, Ariz.
 OLDFIELD, T. W. Care Cinco Mines Co., Magdalena, Jal., Mexico.
 ORMSBEE, JAMES J. P. O. Box 681, El Paso, Texas.
 ORR, JOHN F. Care Chalmers & Williams, Chicago Heights, Ill.
 PACKARD, HENRY J. Stanford University, Cal.

- PALMER, C. H., JR., Min. Engr., Pacific Mines Corp.,
United Eastern Mining Co., Los Angeles, Cal.
- PARSONS, ARTHUR B. 160 10th East St., Salt Lake City, Utah.
- PENBERTHY, JOHN E. Supt., Shannon Copper Co., Gleeson, Ariz.
- PERRY, WILLIAM C. 605 R. A. Long Building, Kansas City, Mo.
- PETERSON, FRANK. 823 S. Bonnie Brae St., Los Angeles, Cal.
- PIERSE, EDWIN A. Fergus Mining Co., Great Falls, Mont.
- POCOCK, CECIL W. 126 W. 91st St., New York, N. Y.
- POMEROY, WILLIAM A. 947 Waverly St., Palo Alto, Cal.
- POTUGAL, J. L., Mgr., Bull Valley Mines Leasing Co.,
Bull Valley, Washington Co., Utah.
- PULLEN, ERNEST F. Alexo Nickel Mine, Porquis Junction, Ont., Canada.
- RAINSFORD, RALPH S. Care General Motors Co., Detroit, Mich.
- REDFEARN, ALBERT M. 146 Belmont Ave., Detroit, Mich.
- REECE, P. P. Supt., Ogden Consolidated Coal Co., Ogden, Iowa.
- RENO, CHARLES A. 717 Lake St., Los Angeles, Cal.
- RINGLUND, S. Kelly, N. M.
- ROBERTSON, WILLIAM A. East Dedham, Mass.
- ROCCA, ANDREW, JR. Helen Quicksilver Mine, Middletown, Cal.
- RODDEWIG, GEORGE W., Engr., Care Federal Mining & Smelting Co., Wallace, Idaho.
- RODGERS, SELDEN S. Washoe Reduction Works, Anaconda, Mont.
- ROSS, JAMES G. Instructed to hold all mail.
- ROYCE, WARD. Hancock, Mich.
- RUSSEL, H. Y., Mgr., Technical Division, Canadian Explosives, Ltd.,
Westmount, Montreal, Canada.
- SAHLIN, AXEL, Care International Engineering Co., Prudential Bldg.,
Sheffield, England.
- SAWYER, A. H. Box 487, Birmingham, Ala.
- SCHIFFNER, C. Schulgasse 10, Freiberg, Saxony, Germany.
- SCHINDLER, D. F. 414 N. Charter St., Madison, Wis.
- SCHNEIDER, GEORGE W., Care T. E. Watters, Suite 529-534, Symes Bldg.,
Denver, Colo.
- SCOTT, O. N. Care Ontario Club, Wellington and Jordan Sts., Toronto, Canada.
- SEARS, MORTIMER A., Mineral Inspector, U. S. General Land Office,
Dept. of the Interior, Denver, Colo.
- SEIDEL, VICTOR B. Apartado 1149, Havana, Cuba.
- SEYMOUR, R. D. 504 Dooly Bldg., Salt Lake City, Utah.
- SIMON, TREVOR B. 1322 E. 112th St., Cleveland, Ohio.
- SKOGMARK, JOHN. Supt., Prime Western Spelter Co., Tiltonville, Ohio.
- SMITH, C. ALBERT. Istmina, via Buenaventura, Colombia.
- SMITH, LYON. Met., Pittsburg Silver Peak Gold Min. Co., Blair, Nev.
- STOCKTON, ROBERT S., Supt., Operation and Maintenance, Dept. Natural Resources,
Western Section C. P. Ry. Irrigation Block, Strathmore, Alba., Canada.
- STRAUSS, LESTER W. Casilla 514, Valparaiso, Chile.
- STURGIS, EDWARD B. Pleasantville, N. Y.
- SWEENT, H. P. Hawksbury, C. B., Nova Scotia, Canada.
- TAYLOR, WILLIAM H. 2725 Whitehall Bldg., 17 Battery Place, New York, N. Y.
- THOMSON, S. C. 14 Wall St., New York, N. Y.
- THORNE, W. E., Lenskoie Gold Mining Co., Nadezhdinsky Mine,
Bodaibo, Irkutsk Government, Siberia.
- TONG, S. K. 412 West 115th St., New York, N. Y.
- TRATMAN, E. E. R. 202 Ellis Ave., Wheaton, Ill.
- TRAUERMAN, CARL J., Min. Engr., Care Rothwell-Trauerman,
832 Colorado St., Butte, Mont.
- TURNER, HENRY W. 634 Mills Bldg., San Francisco, Cal.
- UNDERWOOD, A. J., Min. Engr.; Mgr., Sierra Mining Co.,
850 S. Marengo Ave., Pasadena, Cal.
- VAN ARSDALE, WILLIAM H. Glenwood Inn, Riverside, Cal.
- VAN DORP, G. H. Kappa Sigma House, Colo. School of Mines, Golden, Colo.
- WADDELL, G. F. 77 O St., Salt Lake City, Utah.
- WANG, C. Y., Han-yeh-ping Iron & Coal Co., Ltd., Tayeh Iron Mines, Tayeh, China.
- WARD, M. H. Mgr., Stanley Motor Car Co., 5883-85 Delmar Ave., St. Louis, Mo.
- WEAVER, A. HURST. 203 Jacoby St., Norristown, Pa.
- WEED, WALTER HARVEY. Room 302, 29 Broadway, New York, N. Y.
- WEEKS, WALTER S., Instr. in Min. and Met., Harvard University, Cambridge, Mass.

WENDLER, HERMAN J.	Genl. Mgr., International Investment Syndicate, 523 Los Angeles Investment Bldg., Los Angeles, Cal.
WHITE, J. L.	Supt., De Soto Mine, Cons. Arizona Smelting Co., Turkey Creek, Ariz.
WHITE, JAMES B.	121 S. Galt Ave., Louisville, Ky.
WHITFIELD, H. E.	61 Thomas St., Subiaco, Western Australia.
WIGGIN, ALBERT E.	609 Locust St., Anaconda, Mont.
WILFLEY, C. R.	Ouray, Colo.
WILSON, ELWOOD J.	Apartment 66, 540 W. 122nd St., New York, N. Y.
WILSON, M. O.	14 Wall Street, New York, N. Y.
WITHERELL, C. S.	American Smelting Co., 165 Broadway, New York, N. Y.
WONG, WILLIAM A.	Hanyang Iron & Steel Works, Hanyang, China.
YOUNG, C. M.	Hiram, Ohio.
ZACH, LOUIS M.	Construction Engr., Doe Run Lead Co., Rivermines, Mo.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
GOEDICKE, CARL	Box 535, San Antonio, Texas.
GOODLAND, GILMORE	17 Gracechurch St., London, E. C., England.
GORDON-FIREBRACE, W. E.	812 Salisbury House, London, E. C., England.
HYDE, JAMES M.	1041 Shattuck Ave., Berkeley, Cal.
JONES, EDWARD B.	Box 502, Salt Lake City, Utah.
LAVERY, V. M.	Box 85, Llanerch, Pa.
NEWBERRY, R. W.	Box 1477, Bisbee, Ariz.
PETERSON, FRANK	622 I. N. Van Nuys Bldg., Los Angeles, Cal.
PORTER, ROBERT S.	Care Fortifications, Culebra, Canal Zone.
STEEL, DONALD	202 Emerson St., Palo Alto, Cal.
WEAVER, ALFRED H.	Spanish Amer. Iron Co., Santiago de Cuba, Cuba.
WHITE, R. T.	Dzansoul, Russia.
WRIGHT, PERCY E.	Seattle, Wash.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Jan. 10 to Feb. 10, 1915;

Date of Election	Name	Date of Decease
1899	†† Cooper, James B.	February 27, 1914
1905	* Miller, George L.	August 12, 1914
1907	* Peck, Francis J.	May 17, 1914
1901	* Willmott, A. B.	May 8, 1914
	* Member.	† Life Associate

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

L. W. FRANCIS, *Chairman*, WILLARD S. MORSE, *Vice-Chairman*.
 THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.
 P. A. MOSMAN, *Treasurer*.
 LOUIS D. HUNTOON. WILLIAM A. POMEROY.

Boston

HENRY L. SMYTH, *Chairman*. ALFRED C. LANE, *Vice-Chairman*.
 AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.
 ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

FRANK A. ROSS, *Chairman*. RUSH J. WHITE, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

I. F. LAUCKS, *Chairman*. J. F. MENZIES, *Vice-Chairman*.
 GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
 W. C. BUTLER, H. L. MANLEY.

Southern California

THEODORE B. COMSTOCK, *Chairman*. SEELEY W. MUDD, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 300 Germain Bldg., Los Angeles, Cal.
 A. B. CARPENTER, C. COLCOCK JONES.

Colorado

CHARLES A. CHASE, *Chairman*. S. A. IONIDES, *Vice-Chairman*.
 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*. JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICH LAIST, W. C. SIDERFIN.

San Francisco

G. HOWELL CLEVINGER, *Chairman*. C. W. MERRILL, *Vice-Chairman*.
 JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
 F. W. BRADLEY, ANDREW C. LAWSON.

Pennsylvania Anthracite Section

R. V. NORRIS, *Chairman*.
 CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
 EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
 CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

HERBERT A. WHEELER, *Chairman*. FIRMIN V. DESLOGE, *Vice-Chairman*.
 WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 JAMES MALCOLMSON, R. A. BULL, PHILIP N. MOORE.

Chicago

ROBERT W. HUNT, *Chairman*. J. A. EDE, *Vice-Chairman*.
 H. W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.
 F. K. COPELAND, G. M. DAVIDSON.

Utah

R. C. GEMMELL, *Chairman*. C. W. WHITLEY, *Vice-Chairman*.
 ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
 WALTER FITCH, G. W. RITER.

STANDING COMMITTEES

*Executive*WILLIAM L. SAUNDERS, *Chairman*GEORGE D. BARRON,
SIDNEY J. JENNINGS,JOSEPH W. RICHARDS,
BENJAMIN B. THAYER.*Membership*JOHN H. JANEWAY, *Chairman*KARL EILERS,
LEWIS W. FRANCIS,LOUIS D. HUNTOON,
THOMAS H. LEGGETT.*Finance*CHARLES F. RAND, *Chairman*.

GEORGE D. BARRON,

ALBERT R. LEDOUX.

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DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Enlarging the Worth of the Worker and the Perspective of the Employer

BY J. PARKE CHANNING,* NEW YORK, N. Y.

(New York Meeting, February, 1915)

THESE days of great industrial and social problems in America produce many suggested solutions and great changes. The practical engineer and employer of labor views these problems differently from the labor leader or the social reformer, but as never before he is sincerely interested in solving them in a way that will be just to all.

The inevitable tendency of the day is toward "industrial betterment," "safety," "industrial education," "efficiency," and the many other things which have become so familiar to progressive employers. There is no longer any question that these things are worth while from both the human and economic standpoints. They "pay" in dollars and cents.

The very center of final success in improving conditions and increasing the efficiency of workingmen must be the spirit of fairness and a knowledge on the part of the employer of how to deal *sympathetically* and *intelligently* with his employees. Every progressive employer knows how greatly he desires foremen, superintendents, managers and others who possess these qualities. On the other hand, we are all familiar with serious mistakes made by young graduates of engineering schools who have had no opportunity to develop these qualities, and who have no real appreciation of the worth of the workers. Indeed, one wonders whether much ill-feeling, labor difficulties, and many strikes could not be avoided if such men had the right attitude.

Is there any way of remedying this condition? If this particular difficulty can be solved, if these young engineers, many of whom are our coming leaders of industry, can be given the right perspective and the right understanding of these other problems in addition to fair, sympathetic methods of handling men, many of our other problems will be solved—not at once, but gradually and permanently, as these men make good and become influential in paths of industrial righteousness and industrial peace. Many progressive employers of to-day have enlarged their own perspective and realize the great importance of enlarging the perspective of those who shall follow them.

How can it be done? For seven years a movement has been making

* Vice-President, Miami Copper Co.

rapid progress in engineering schools with the purpose of helping to solve this very problem. It was started at Yale in 1907, by the Young Men's Christian Association, when some engineering students were led to get in touch with workingmen and boys in New Haven. The idea was to render service by teaching them English and other subjects and in turn to learn their ways, ideas, customs, and how to deal with them intelligently. Friendly, mutually helpful personal contact was the basic principle. This was the beginning. Do not confuse it with "social service"—it was this, and much more. The reaction on the engineer was the main object sought. The idea worked out so successfully that a number of men saw great possibilities in it, and the whole conception was greatly enlarged. Under the name of the Industrial Service Movement, it has spread to 200 other colleges and technical schools in the past seven years, and has justified itself from every point of view. It is really helping in a vital way to solve the special problem we have been discussing and other problems as well. It is to put it briefly:

Plan: Bringing engineering students and industrial workers together to their mutual understanding and their mutual good.

Purpose: To get workingmen educated and educated men to work. To send men out of college with a new sympathy, a new vision and a new determination to help.

Principle: Fraternity—not to go down to help others or to ask others to come up and be helped, but rather to go *with* them, not in any sentimental way, but in a spirit of common-sense brotherhood.

Method: Putting college students up against real opportunities for the kind of service which appeals to them, such as teaching foreigners English and citizenship; instructing American workingmen in technical subjects; leading clubs of working boys, etc. There is opportunity for every leader's peculiar ability to assert itself, in any way that is real. Other methods will be described later.

Accomplishment: During the past year 3,500 students from 200 colleges have engaged regularly in industrial service; 3,000 graduates are active in industrial betterment as a result of interest acquired while at college, during the past seven years.

Leadership: The Young Men's Christian Association, through local branches, State committees and the industrial and student departments of the International Committee.

Co-operation: The Movement works locally through the Young Men's Christian Associations and any other recognized agencies for industrial and social betterment in the community. Professors and students, employers and employees, engineers and social workers heartily co-operate.

Significance: Experience proves that men interested in this work at college go out into the larger world with a new vision and a new attitude and sense of responsibility. These men will largely determine whether



WANTED: LEADERSHIP.

conditions shall be good or bad and whether the human factor will be given fair consideration. How better can the problems of capital and labor be solved than by mutuality, good will, efficiency and character in business? The nation's hope is in the coming leaders who shall follow us and who possess such essential qualities of success. The development of such leaders, with their continually increasing capacity for service, is the ultimate purpose of the Industrial Service Movement.

It may seem surprising that 3,500 engineering students, each carrying a heavy course of study and with many other interests, can find an evening



SHOP MEETING.

or two each week to engage in some form of definite service, without any financial compensation. But such is the case, and on the whole a careful survey of their work reveals efficiency and permanency in a high degree. If industrial men are at first suspicious, their suspicion soon wears away in the face of frankness and friendliness. If the employer has any doubts, they do not last long. One may travel around the country and observe students teaching foreigners in railroad box cars, stores, clubs, halls, pool rooms, restaurants, and boarding houses as well as in the more dignified meeting places—schools, churches, settlement houses, and factories. One may see American workingmen instructed in mines, shops, and labor-

union headquarters. One may look with interest upon recreative games, talks, first-aid and safety promotion in all sorts of places at noon, afternoon, evening, and midnight. And one may see 500 men crowded around the machinery of a huge plant listening to a straight noon-hour talk on clean living, character-building, and vital religion. We have looked with amazement on 50 factory boys following enthusiastically a college football captain who took enough interest in them to organize a boys' club or a factory athletic league. It has all been done in the finest kind of spirit, without patronage, with modesty and with efficiency. And during the past year those 3,500 student leaders reached over 60,000 workingmen and boys in a very personal and directly helpful way. The Secretary of this Movement has talked with hundreds of employers and college professors throughout the country and all seem enthusiastic over what has been accomplished.

But what has this to do with engineering? Just this—that every one of these 3,500 students would be willing to say that he has gained far more than he has given. Furthermore, a study of the situation proves that he has gained in large measure the very qualities he needs—an appreciation of workingmen, adaptability, leadership, a knowledge of how to deal with men in a way to get results and to avoid harmful labor difficulties. In general, he learns that all men are men, regardless of race, nationality, color, or creed, but that men must be dealt with very differently; he learns that it pays to win the leaders of men if one desires to win the men themselves; that the work, home and leisure life of industrial workers play a large part in determining efficiency; that a man's working associates may largely influence the quality of work he does; that helping men to concentrate on their work (though not at the expense of mental and physical welfare) increases output; that friendly competition (without driving men) helps break records; that reasonable relaxation and recreation pays both from the human and economic standpoints; that visitation of other plants and stimulation of new ideas in various ways may mean a money saving to the company; that loyalty of the men is one the employer's greatest assets; and that character counts most of all. More than this, he learns to understand men, he learns how to sympathize with the other fellow's point of view and how to handle men successfully. Is this not worth while? Who can foresee what the future will hold for these men in the way of tremendous opportunities and responsibilities?

Let us illustrate. One engineering student apparently never took any interest in any one but himself until he was enlisted in some of this work. Two evenings each week he walked two miles to teach a class of 20 coal miners. The miners learned a great deal, but they little realized how much they were teaching the college man. His whole viewpoint was gradually changed. He learned to appreciate that all men were *men*, and he graduated from college with a new vision and a new sense of responsi-



HEALTH TALK TO PITTSBURGH STEEL WORKERS AND FAMILIES.

bility. He had not become a sentimental idealist. He perhaps realized the weaknesses of workingmen better than ever, but he had come to know their good points as well, and he had developed a real point of contact. It was therefore not surprising to receive a letter from him recently, indicating his growing interest and telling enthusiastically of his success with a welfare club house and other educational, recreative, and social features introduced for his miners.

It happens that I am a member of the Advisory Committee of the Industrial Service Movement, the headquarters of which is at the office of the Y.M.C.A. International Committee, 124 East 28th Street, New York. Several times I have seen letters from recent engineering graduates, which have come to the central office. These letters tell the story in no uncertain terms, and from them the following quotations are taken:

"I have organized several classes for the men in our plant. I can say honestly that this friendly basis with my men helps rather than hurts discipline. The work is but a beginning of what my company hopes to do."

"You will no doubt remember that I took up this work last winter at the University. Now that I have gotten into the habit I really like it so much that I am devoting some of my time to it now, though I am very busy with my business. I have gathered together a class of about forty Italians and enjoy it immensely."

"My company is just now organizing a scheme whereby classes will be offered to young men in such subjects as relate to their work. I have consented to teach one of these groups, as this is the same sort of work I did when at college."

"I have read the literature with care and interest. I expect in the near future to be called to a position in New Mexico where I will come into close contact with foreign miners. My special work will be in the promotion of education and accident prevention."

"I consider the contact that I had with the Industrial Service Movement the most valuable experience in my undergraduate days. I would never be where I am to-day, without it."

There are numerous other ways in which the Industrial Service Movement is making itself felt in the colleges. These can only be mentioned briefly. Special lectures on the human side of engineering and kindred topics are given by selected employers, engineers, and social workers. These give the students a vision which does not come from most technical courses. Some colleges have promoted strong two-day conferences on safety and industrial betterment. The student engineering societies are stimulated to have such lectures and to take up the practical aspects of the work. Weekly discussion groups are held for interested students an industrial reference library is installed, and a live bulletin board keeps the work before the student body. As senior engineering trips pass through New York, Chicago, or other cities where the Movement is strong, professors in charge frequently arrange for a day to be spent in studying "the human side" and the work of various welfare agencies. One of the most significant recent developments is the proposed course on the "Hu-



CLASS IN AMERICAN CITIZENSHIP.

man Side of Engineering," which will be hereafter included in the regular engineering curriculum of some of the largest colleges. This course will include such topics as: "The Human Factor in Industry," "Historical—The Evolution of the Individual Worker," "Industrial Organization," "Influence of the Modern Factory System," "Evil Conditions which Retard Progress," "Efficiency Conditions," "The Welfare Programs of Typical Companies," "Industrial Betterment," "Scientific Management on its Human Side," "How to Handle Men," and "The Engineer's Responsibility for Service."

Many articles about the Movement have appeared in engineering and other magazines. Some of the national engineering societies are very much interested in the whole plan, and many points of co-operation between the American Institute of Mining Engineers and this Movement will at once be apparent.

There is another side of this whole question which must not be forgotten. The Movement stimulates great interest in the colleges. While 3,500 students are actually engaged in service, many thousands of other students and professors hear the Movement presented and become heartily interested in its ideas and ideals. Is there danger of this enthusiasm not being conserved after graduation?

The Young Men's Christian Association recognizes this very danger, and willingly places its world-wide organization of over 8,000 branches at the disposal of these engineers who have become interested in the service idea. Whether the graduate engineer goes to the frontier or into the industrial city, this Association is an instrument ready for his use. In the larger cities it is being used to perform definite pieces of service which the industry may need. Industrial and immigration secretaries are available to co-operate. For example: A recent graduate entered the employ of the Ford Motor Car Co., in Detroit, and at once manifested his interest in the foreign employees. The company officials noted his enthusiastic interest, and used him in their enlarging plans for educating their men. He naturally turned to the Young Men's Christian Association, which had prompted his interest while at college, and brought the Association's special methods of work with foreigners to the attention of the production manager. The result is that over 1,100 men in that plant are now learning English, and American citizenship, under the auspices of the Association. Special lessons were prepared for this industry. This is being done also for the mining industry, and accident prevention is taught with the lessons by graphic pictures. In the remote engineering fields this co-operative agency is quite as valuable as in mining and lumber centers and in construction and reclamation camps. Thus the machinery is available, and members of this Institute will be especially interested in what some mining men have said of it.

Dr. J. A. Holmes, of the U. S. Bureau of Mines, says:

"I am interested in the Y.M.C.A. because during the last twenty-five years, as I have passed up and down through the great mining regions, I have seen places transformed through its work."

At a dinner recently, John Hays Hammond made the remarkable statement:

"This work is one of the most promising things that I have yet seen in connection with modern industry."

Judge Elbert H. Gary, Chairman, Board of Directors, United States Steel Corporation, has said:

"I am glad to declare my belief in the advantages of having a Young Men's Christian Association in an industrial community as tending greatly to the building up of the character of the men, and therefore increasing their efficiency."

It is needless to say how much I am personally interested in the movement. The principle is a sound one, and I can already, from my own experience, see the good effect it has not only upon the young men who are engaged in this work but upon the men to whom they are devoting their energies.

Any such agency which can improve the educational, physical, social, and spiritual welfare of employees is bound to increase their efficiency. Fred H. Rindge, Jr., Secretary of the Movement, writing in the *Mining and Scientific Press* of Nov. 7, 1914, said:

"Practical money-making industrial leaders (even the unsentimental ones) have found that such a factor in modern industry develops trustworthiness, higher standards of workmanship, and makes possible that kind of intercourse between employers and employees which begets good will, and good will raises all other forces in industry to a higher power. The right kind of an investment in men will yield big returns."

The whole plan seems to me sane, practical, and convincing. It helps the coming engineer to get a broader perspective, while an undergraduate. It enlarges the worth of the workers through the service of the students. It affords an instrument for all-round welfare work wherever the engineer may go. I am reading this paper because I believe our Institute will want to know more of this work and because of the direct opportunities for helpful co-operation through the Affiliated Student Societies and through our members at large. Think of what it will mean to industry to have a thousand or more men each year entering upon their work with this splendid spirit, and with the ability to inspire confidence, good will, and loyalty. What will this mean to the industrial leadership of the future?

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

High Blast Heats in Mesaba Practice

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(New York Meeting, February, 1915)

INTRODUCTION

THE use of high blast heats on furnaces melting Mesaba ores is still the exception, the average blast temperatures carried on Mesaba stacks seldom reaching 1,100° F. Some 15 years ago, when the use of fine Mesaba ores in larger proportions to the total burden first came into practice, it was found difficult to employ even those blast heats which it had been possible to carry with the Old Range ores. The furnaces refused to "take" high heats, and would persistently labor with irregular stock movement, which impaired the practice and increased the coke consumption. By lowering the blast temperature, the working of the furnace became smoother, and experience taught that with the softer and more easily reduced Mesaba ores, satisfactory practice could be obtained with comparatively low blast heats. Therefore a large number of Mesaba furnaces have ever since been operated on very low blast temperatures, ranging between 800° and 1,000° F. In the face of these difficulties, furnace men were slow to recognize that furnaces could be so designed as to permit a free movement of the charges in combination with the use of higher heats. Since higher blast temperatures did not seem to offer great advantages in the smelting of Mesaba ores, investments in hot-blast stoves were not as attractive to the furnace man as labor-saving devices and means of obtaining greater tonnages.

The effect of this preference is keenly felt at the present time. It has brought about the peculiar situation, that the majority of our furnace plants (among them some of the most progressive) lack modern and efficient stove equipment. The result is that, even where the proper furnace lines have been adopted and conditions are now favorable to the successful employment of higher blast temperatures, the heats required for the greatest fuel economy are not available.

In an endeavor to obtain results, even with inadequate equipment, we find that, at many plants, the stoves are worked far beyond their capacity. This leads, in many instances, to a situation in which the coke saving in the

furnace is offset by the losses of heat carried away in the waste gases escaping from the overburdened stoves. Quite frequently, 40 per cent. and more of the total gas production is used in heating the blast, thereby seriously curtailing the amount of gas available for conversion into power. The more efficiently this power is generated, and the greater the demand for it, the higher becomes the value of the furnace gases, and the more necessary it is to economize the gas used in the stoves.

There are furnace plants which have raised their blast temperature to an average of 1,000° to 1,100° F. and are obtaining very low fuel consumption with Mesaba ores, by means of good furnace lines, suitable coke, correct distribution, and other essentials. From considerations of the high value of the surplus gas on the one hand, and lowered stove efficiency under forced operation, on the other, it may have appeared uneconomical to attempt higher blast heats, quite apart from the theory according to which the fuel saving to be expected by raising the blast temperature above a fixed range varies inversely with the reducibility of the ore burden, and rapidly decreases with increased temperatures. Accordingly, occasional attempts to carry still higher heats were, as a rule, abandoned.

But actual practice in recent years has proved a different conclusion, having shown that a considerable coke saving is still obtainable when the heats are raised above the 1,100° line. I make this assertion, notwithstanding the fact that some experiences, under conditions unfavorable to the use of high heats—conditions such as are inherent in raw materials, furnace lines, and methods of operation—may have tended to indicate the contrary.

In confirmation of this point I may mention that, during the last year, the blast temperatures at two of the Bessemer furnaces at the South Chicago plant of the Illinois Steel Co. were raised above the 1,100° line, and for many months temperatures from 1,100° to 1,300° F. were employed, with the result that the coke consumption was reduced to extraordinarily low figures without causing any operating difficulties. Data from the practice of these furnaces are shown in Table I.

Since it has been shown, therefore, that with proper furnace conditions it is not only possible, but profitable in coke economy, to carry high heats on Mesaba furnaces, the next problem is to generate these heats in the most efficient way, and without losing through inefficient stove operation the advantage thus to be gained. In other words, the problem of adapting furnace lines, raw materials, and operation to the new practice having been proved capable of solution, efficient heat generation, through suitable stove construction and operation, is of paramount interest to-day.

STOVE CONSTRUCTION

It is not my intention to comment here on the various types of hot-blast stoves, or to discuss the advantages of any special design. I shall give

TABLE I.—Data Concerning the Performance of No. 4 and "E" Blast Furnaces at South Works, Illinois Steel Co.

Date	Average Daily Product	Pounds per ton					Per cent. Mesaba	Cu. Ft. of Wind per Min. at Engines	Average Blast		Iron Analysis		Actual Yield
		Ore, Scale, Etc.	Coke	Stone	Scrap Used Over Product.	Flue Dust Produced			Temp.	Press.	Si.	S.	
1914		Blast Furnace No. 4.											
*Feb....	290	4,572	2,464	1,029	-54	480	62.6	29,600	1,085	11.1	1.56	0.031	50.05
Mar....	518	4,062	1,744	781	17	193	67.4	41,140	1,170	15.7	1.29	0.025	54.76
Apr....	543	4,003	1,702	752	73	227	66.5	42,950	1,196	16.6	1.23	0.027	54.40
May....	530	4,076	1,733	785	12	152	61.4	46,260	1,161	15.4	1.24	0.028	54.68
June....	572	4,099	1,699	804	13	82	94.2	48,970	1,243	15.6	1.35	0.032	54.36
July....	560	4,196	1,765	858	-13	109	95.8	47,670	1,164	15.4	1.33	0.032	53.65
Aug....	549	4,190	1,828	884	-23	81	96.4	44,950	1,129	15.1	1.37	0.033	53.91
1913		Blast Furnace E											
Sept....	525	4,012	1,691	677	118	169	92.9	47,750	1,225	15.1	1.30	0.034	53.24
Oct....	514	4,047	1,711	774	113	171	93.0	46,390	1,251	16.2	1.48	0.043	52.85
Nov....	467	3,850	1,783	827	131	238	82.7	46,790	1,157	16.4	1.51	0.039	55.22
†Dec....	536	3,852	1,705	780	109	201	83.7	45,510	1,250	15.7	1.33	0.041	55.69
1914													
†Jan....	463	4,067	1,821	819	55	299	82.9	43,350	1,230	15.4	1.29	0.038	53.82
Feb....	489	4,216	1,850	827	11	245	74.4	42,940	1,200	16.6	1.42	0.049	52.82
Mar....	559	4,144	1,815	790	18	197	66.3	44,120	1,113	17.9	1.31	0.037	53.63
Apr....	582	4,035	1,742	775	23	234	65.9	43,950	1,146	17.7	1.27	0.028	54.92
May....	538	4,205	1,890	868	18	161	51.5	45,740	1,059	16.4	1.43	0.029	52.87
June....	558	3,993	1,894	856	22	81	78.7	46,190	1,009	15.4	1.40	0.030	55.61
July....	548	4,073	1,876	875	-22	108	95.5	46,040	1,128	15.7	1.35	0.036	55.47
Aug....	520	4,061	1,981	893	-21	79	96.2	45,680	1,098	14.8	1.45	0.036	55.60

* Blown in, Feb. 16, 1914. Tonnage produced on lining to Sept. 1, 104,350 tons. Pounds of coke per ton of product, 1,773. Average phosphorus, 0.075. Average manganese, 0.60.

† Blanked, Dec. 24, 1913, to Jan. 10, 1914, inclusive. Blown in, Feb. 11, 1912. Tonnage produced on lining to Sept. 1, 448, 139. Pounds of coke per ton of product, 1,933. Average phosphorus, 0.074. Average manganese, 0.57.

only a general outline of the conditions essential for economically successful stove practice.

When at an existing furnace plant the use of higher blast temperatures is contemplated, the simplest means is often deemed to be an enlargement of the stove capacity by building an additional stove. This will, of course, increase the available heating surface and produce a higher blast temperature, but not without considerably decreasing the efficiency of the hot-blast equipment. Besides complicating operation, by increasing the number of valves and pipes, it adds to the radiating surface of the stoves, and thereby to the radiation losses; for instance, where a fifth stove has been erected, these will increase at least in direct proportion, *i.e.*, 25 per cent.

When generating higher blast heats in the above manner, the average temperature of each stove necessarily becomes higher. This means a further decrease in efficiency, since a higher stack loss is the unavoidable result. With the average stove temperature the temperature of all the radiating stove surfaces also increases, causing another considerable loss of heat, which is better realized when it is remembered that radiation increases as the fourth power of the temperature.

In spite of all these disadvantages, many blast-furnace managers have decided that the erection of a fifth stove is the best means of increasing their stove-capacity, for two obvious reasons: (1) because it can often be done with practically no other changes to existing equipment; and (2) because it insures a greater regularity of blast temperature.

For instance, where five stoves are available, it is possible periodically to cool off each unit completely for cleaning the checker work, without any loss of heat to the blast furnace. Between cleaning periods, two stoves can be operated on wind simultaneously, and three stoves on gas, which method permits a constant blast temperature to be maintained, at a higher level and with less stove changes than could be obtained by by-passing cold blast into the hot-blast main through a mixer valve. It is also claimed for this practice that each stove can be cooled, without decreasing the blast temperature on the furnace, considerably lower than is otherwise possible; and that thus the heat interchange from gas to checker work and from checker work to blast becomes more active, resulting in lower stack temperatures and better efficiency. This may be true, as long as this method is compared with ordinary practice on the same stoves, and each stove is considered individually. But it does not hold true for the set of stoves as a whole, the heat-receiving and forwarding capacity of which is really reduced to but little over that of a four-unit stove plant, while it suffers all of the radiation losses of a five-unit plant.

That the possibility of periodically cleaning each stove without heat losses to the furnace is a considerable advantage over the old

practice is not disputed. Still, the five-stove practice, like any practice with dirty gas, entails the tremendous disadvantage that each stove has to be completely cooled off when its turn for cleaning comes. The working periods between these cleaning operations can be lengthened by the well-known method of blowing the checkers off with steam; but it is impossible thus to eliminate the trouble entirely. Each time a stove is cooled off, its total heat content is lost; but the more serious objection is the contraction of the brick work throughout the stove, and its re-expansion during the following period of heating, which opens the joints and causes the brick to crack, the checker work to shift, and the supporting arches to crumble. In many instances these disturbances lead to the actual breaking down of the brick work, and always hasten the final destruction of the lining.

Mechanical cleaning, too, injures the stove lining, especially the top checkers and the combustion chamber, where the flue dust carried in with the gas sinters, often melts and fuses with the brick work.

While it appears that having a fifth stove available is beneficial to the furnace practice, in allowing the regular maintenance of higher heats, yet this benefit is gained at the expense of stove efficiency, because the chief evil has not been removed, namely, the accumulation of flue dust in the stove checkers, which constitutes the greatest impediment to efficiency in stove practice. The only way to eliminate this definitely is to prevent the flue dust from entering the stoves, *i.e.*, to wash the gas thoroughly. It can be considered an established fact that it is impossible to free the gas from its dust content sufficiently by dry-cleaning systems alone, except by filtering. The latter process seems to be rapidly gaining prominence in localities where the water supply is limited. In the majority of cases the washing of stove and boiler gas in towers or rotary machines seems to be the most desirable method from an operating as well as economical point of view.

In Mesaba practice, the washing of the gas is specially important, and it is incomparably more beneficial than in furnaces melting hard ores. This is mainly for two reasons. By reason of the fineness of the ores and the fast rate of driving customary at Mesaba stacks, the amount of flue dust produced is high, and most of it is a very fine powder, which cannot be eliminated from the gas by any existing dust catcher. Hence the clogging of the checker openings proceeds very rapidly in Mesaba practice. On the other hand, the top temperature of Mesaba furnaces being comparatively low, the loss of sensible heat, which always accompanies the washing of the gas, is reduced to a minimum, and is offset many times by the elimination of the water vapor and the resulting concentration of the combustible constituents of the comparatively lean gas.

But while, in Mesaba practice, conditions without exception favor the use of washed gas, there may be considerable room for argument to

the contrary in the case of furnaces working on coarser and harder, especially magnetite, ores. The flue dust carried out by the gases from such furnaces may be small in quantity, and of such a nature as to be easily and sufficiently eliminated in the dust catcher. Furthermore, by reason of the high temperature of the rich top gases, the loss of sensible heat through washing becomes serious, and may eventually justify a decision against the installation of a wet-washing arrangement.

Notwithstanding this exception, it is the use of washed gas which has made possible the construction of stove linings with a view to stability and heat economy only, and without regard to the consideration of stove cleaning, which, if the gas be dirty, must become the prime factor in determining the checker-work construction. The checker openings, which, where unwashed gas is used, are usually 9 in. square, can, with washed gas, readily be reduced to as small a size as the draft will allow. The checker walls, which, with unwashed gas, are not considered safe when less than 3 in. in thickness, can now be reduced to the limit set by the size of checker openings as determined by the proper relation of total brick volume to surface, and by the ability of the brick manufacturer to produce a strong, substantial firebrick. In this way, the heating surface of the checker work is increased, for instance, more than 50 per cent. by reducing the checker openings from 9 to 4 in., and the thickness of the checker walls from 3 to 2.5 in. At the same time, this construction eliminates the dead portion of the brick work, in the center of the thicker checker walls, which does not participate in the heat exchange, as is proved by the fact that during the ordinary heating and blowing periods it undergoes hardly any change of temperature.

By using washed gas and changing the stove linings along the lines indicated above, any furnace plant can increase the heating capacity of its stove equipment without adding to the radiating surface. In many cases an increase which will fully meet all heat requirements can thus be obtained.

With higher stove temperatures, better insulation of the stove shell gains importance. It can be obtained by making the outer brick walls thicker, and using for them two kinds of brick: a porous kind of highly insulating quality, next to the shell; and, inside of this, a course of common firebrick (except in the combustion chamber, where a lining of highly refractory brick is required). Under no circumstances should the depth of the insulating wall be sacrificed to the desire of increasing the heating surface. To get an idea of the heat saving which can be expected from a better insulation, one has only to consider that in most stove plants the radiation losses amount to fully 20 per cent. of the total heat generated.

If sufficient heating capacity cannot be obtained within shells of a given size, the only remedy is to enlarge them, by increasing either the height or the diameter. In building new stoves a larger diameter should

be chosen; and on existing units an increase in height is generally feasible and sufficient.

STOVE OPERATION

With proper equipment and clean gas, stove operation becomes an entirely different problem. It is no longer largely a question of crowding as much gas and air into the stove as possible. The scientific rules of combustion can now be applied, and the stoves can be heated with a minimum amount of gas by complete combustion, proper draft regulation, and an even distribution of the combustion gases and air over the entire heating surface.

Combustion of Washed Gas

In almost all cases where the change has been made from unwashed to washed gas, difficulties have been encountered, due to the slower ignition of the latter. The cold and apparently wet gas would not ignite opposite the gas and air inlets, and the flame would burn only in the upper parts of the combustion chamber and often during its downward passage through the checker openings. Stoves would look dark and cold at the bottom of the combustion chamber, and, in extreme cases, even a large excess of air would not free the stack gases entirely from carbon monoxide.

Where such difficulties have seriously interfered with the operation, and could not be overcome by adapting the stove practice to the new conditions, their cause has generally been either insufficient washing or inadequate drying of the gas.

The former trouble is manifested by too great a difference between the temperature of the water entering and the gas leaving the washer, and by an excessive deposition of mud in the gas conduits, which, after only a few weeks' run, may clog the gas outlets or burners. The gas, which, after passing through the washers is, of course, always saturated with moisture, contains at the higher temperature a correspondingly greater amount of water vapor, which, together with the active components of the gas, has to be heated up to the ignition temperature before combustion can take place. This vapor, on account of its high specific heat, considerably lowers the temperature of combustion.

Insufficient drying of the gas after washing demonstrates itself in a similar way. The gas current, though of sufficiently low temperature, contains an excessive amount of entrained moisture, which affects the ignition and combustion in a like manner.

Both these conditions cannot be remedied at the stoves and will always injure the stove efficiency by increasing the amount of inert gases passing through the stove during the heating period, decreasing the tem-

perature of combustion and raising the amount and temperature of the waste gases, *i.e.*, the stack loss. The place to eliminate these evils is at their source, the gas-washing and drying plant.

However, it cannot be denied that even with satisfactory washing and drying, it is generally more difficult to achieve quick ignition with cold washed gas than with hot gas. This difficulty, due primarily to the loss of sensible heat, becomes more serious with a decreasing percentage of combustibles in the gas. At plants running on regular low-silicon grades of iron, and with a low coke rate, such conditions may require special attention, and sometimes special devices. To remedy this evil, various arrangements have been proposed, among them the pre-heating of the gas or air, or both, by the waste gases, in heat inter-changers. Nearly all such remedies require a complicated installation, which is undesirable and, under most conditions, unnecessary. I can hardly conceive of a blast-furnace plant working on gas so lean as to make it impossible to get complete and quick combustion simply by intimately mixing gas and air before the ignition takes place. In most cases, a properly constructed burner answering these requirements is all that is needed.

Very efficient burners of a modified Bunsen type have recently been introduced, which, without being complicated in design, give an excellent mixture of air and gas. They utilize the method of first mixing primary air and gas outside of the stove before the point of combustion is reached, and then introducing the secondary air in the proportion required for complete combustion. In this manner a high flame temperature is reached, and all of the carbon monoxide is burned with a much smaller excess of air than is possible with the ordinary type of burner. However, where the size of stoves is entirely inadequate, or relining is not possible in the immediate future, a new method of forced combustion is suggested as the proper solution.

This method, originating in Germany, and now being tried at several American plants, consists of blowing air into the gas burner with sufficient force, so as to burn the mixture of gas and air under pressure within the stove. This not only insures a quicker combustion, but also causes a much greater volume of gas and air to enter the stove than could be drawn in by the chimney draft alone. It is claimed, and experiments tend to show, that in this manner practically as much heat can be stored in the stove in one hour as can be taken out by the blast in the same length of time. If this claim can be established, it is evident that radiation losses will be greatly reduced, since only two stoves in place of the present four would be required. The cost of installing and operating a fan appears to be a very small item, when compared to the first cost, repairs, and maintenance of two hot-blast stoves.

But the installation of a good stove burner is by no means the cure of

all evils. Its proper operation is at least equally important. Weather and draft conditions, the composition and pressure of the gas, and the temperature within the stove, all of which are subject to considerable changes, are vitally influential. Constant watchfulness on the part of the furnace management and crews is required to keep the three variables, gas, air, and draft, always properly adjusted. The importance of strict supervision cannot be over-estimated, and this should be facilitated as much as possible by frequent gas analyses, accessible and easily readable gas-pressure gauges, recording stack-temperature pyrometers, and convenient means of observing the development of the flame in the combustion chamber. The aim should always be to complete the combustion in the lower part of the chamber; the shorter the flame, the higher will be its temperature, and the brighter also the appearance of the combustion chamber walls, which can readily serve as a guide for the operator. If the combustion chamber is roomy enough to permit a free development of the flame, the danger of overheating and melting the parts of the walls opposite the burner is remote.

Distribution of Gases

Proper combustion having been achieved, the next step is to distribute the gases correctly over the entire heating surface, and to bring them into the closest contact with the brick work, there to give off their heat as uniformly and completely as possible.

To facilitate proper gas distribution, many arrangements have been proposed and applied to the original designs of both side and center-combustion stoves. Most of these devices, even if correct in principle, have achieved but little, so long as unwashed gas was used. The deposition of flue dust on top of the checker work would, in the course of a few weeks, bring gas circulation to a complete standstill in a large percentage of the checker openings. Through the remainder of the openings the gas would flow with increased speed, passing only a small part of the total heating surface with little chance for proper heat exchange, and with correspondingly high stack temperature as a result. One can readily see that, under such conditions, any attempts to improve the draft distribution by more adequately spacing and dimensioning the stack valves, by the reduction of the checker openings in such parts of the stove cross-section as are favored by the draft, or by the altering of the dome-shape, would avail little—the deposition of flue dust being, in spite of all, and above all, the predominant factor influencing the draft and the gas distribution.

With the introduction of washed gas, this condition is changed completely. The checker work can now be relied upon to retain its original free area throughout a whole furnace campaign, or at least for a number

of years. The proper distribution of the gas and the correct regulation of the draft now become a problem of the utmost importance. Researches which have been carried on in recent years seem to indicate that on a large number of stoves the deficiency in this respect is serious, affecting in some cases over 50 per cent. of the total heating surface. Such conditions can now be readily and reliably investigated and adjusted. The means to be employed to this end have been indicated above and depend almost entirely on local conditions. At present a large number of furnace managers are giving this subject their special attention; and, since the goal to be attained is well defined, considerable progress in this direction may be expected in the near future.

That the heat transmission to and from the brick work is not impaired any longer by a flue-dust coating on the brick surface, is another important benefit derived from the use of washed gas. In the cooler parts of the stove, this coating appeared in the form of a spongy, porous insulating covering, and in the hotter parts, reacted with the brick work to form an iron silicate glaze of low heat conductivity. How this reaction, penetrating to the interior, deteriorates the brick, making it brittle and easily fusible, I have already observed, remarking also that nothing can be done during operation of the stove to prevent this reaction.

This danger being eliminated, the ability to withstand such chemical attacks is no longer required for a good stove brick; and the manufacturer can now concentrate his efforts exclusively upon the development of a standard quality, which will combine strength and elasticity with highest heat-storing capacity, conductivity, and infusibility.

Great progress along these lines has been made in recent years through the manufacture of machine-pressed brick, the practical service of which is the most convincing argument against the old idea that a checker work must have pores for retaining the heat, instead of the property of taking up and giving off the heat quickly. Such pressed brick will permit a considerable decrease in thickness of the checker walls, without endangering the building strength of the brick work.

Used in conjunction with smaller checker openings, it will enable us to obtain within a given stove shell the maximum heating surface with the largest brick volume and the highest ratio of active brick surface to total free space. This means the most intimate contact between the brick work and gas or air; and, together with concentrated combustion, correct gas and air distribution and draft regulation, will result in the lowest stack temperatures and highest and most uniform blast heats; that is to say, in efficient stove operation.

This paper by no means pretends to give an exhaustive account of all possible improvements which may be applied with more or less success to stove construction and practice. Its aim is solely to awaken an interest in the subject, and invite discussion of the improvements in our

stove practice, particularly with washed gas. I believe that in no other branch of our furnace work will earnest, practical and scientific experimental work be better rewarded.

HEAT CALCULATIONS

The exceedingly favorable practice which has been obtained recently on Mesaba furnaces by the use of higher heats, lends new and special interest to the theoretical side of the fuel-economy question. Research along these lines has so far not been very extensive in Mesaba practice; and it is to be hoped that the recent practical progress will give a new impulse also to scientific investigations.

Examinations of existing furnace conditions in different localities appear to be of first importance; and, as a contribution of this character, a thermal analysis has been prepared of the performance of No. 4 Blast Furnace at the South Works of the Illinois Steel Co., during the month of June, 1914. It is hoped that this attempt will invite discussion and research along similar lines, and thus do its share toward throwing some light on the question, to what extent, under modern conditions, the various reactions in the blast furnace influence the equilibrium between heat supply and demand, and through this the fuel economy.

The data of practice, and the calculations employed for the establishment of the heat balance, follow in detail.

Calculation of Operating Data

Analysis of Top Gases

	By Volume	By Weight
CO ₂	14.9	22.3
CO.....	23.5	22.4
(1) H ₂	4.1	0.3
CH ₄	0.2	0.1
N ₂	57.3	54.9

On the basis of above data one ton of this gas contains:

$$(2) \quad \frac{22.3 \times 3}{100 \times 11} = 0.0608 \text{ ton of carbon as CO}_2$$

$$\text{and} \quad \frac{22.4 \times 3}{100 \times 7} = 0.0960 \text{ ton of carbon as CO}$$

This carbon originates from:

(a) The carbon charged, minus the amount of carbon transferred to the iron and the amount carried off with the flue dust.

(b) The CO₂ content of the raw materials.

These items are calculated below:

(a) The total amount of coke charged during the period under consideration was

(3) 13,006 tons,

which analyzed:

	Fixed Carbon	Ash	S	P	Volatile Matter
(4)	89.00	9.40	0.59	0.008	1.35 per cent.

(5) The coke as weighed averaged 2.30 per cent. moisture, making the weight of dry coke charged:

(6) 12,707 tons.

The amount of fixed carbon charged is thus:

(7)
$$\frac{12,707 \times 89.00}{100} = 11,310 \text{ tons.}$$

From this total is to be deducted:

(a1) The carbon carried off with the iron, which is calculated as follows:

(8) The carbon content of the metal was 4.20 per cent. and

(9) 17,146 tons of iron were produced, besides

(10) 173 tons of scrap (ladle skullings and pig-machine scrap); therefore

(11) 17,319 tons of iron (17,146 + 173) carried off

(12)
$$\frac{4.20 \times 17,319}{100} = 727 \text{ tons of carbon.}$$

Besides the regular burden, the furnace remelted 276 tons of scrap iron, which contained:

(13)
$$\frac{276 \times 4.20}{100} = 11 \text{ tons of carbon.}$$

Therefore the net amount of coke carbon carried off with the metallic product is:

(14)
$$727 - 11 = 716 \text{ tons.}$$

(a2) During the month 628 tons of flue dust were taken out of the dust catcher of the furnace. Assuming that the gas carried over an additional 10 per cent. of this beyond the dust catcher, the total amount of flue dust produced by the furnace was,

(15) 691 tons.

(16) Since the average carbon content of the flue dust was 8.25 per cent., the amount of carbon carried from the furnace with the dust amounted to

(17)
$$\frac{8.25 \times 691}{100} = 57 \text{ tons.}$$

Consequently the net amount of carbon gasified in the furnace is:

(18)
$$11,310 - (716 + 57) = 10,537 \text{ tons.}$$

(b) The only burden constituent containing CO₂ in larger quantities was limestone, of which 6,154 tons were charged. It averaged 43.1 per cent. CO₂,

(20) making the amount of CO₂ charged with the limestone
$$\frac{43.1 \times 6,154}{100} = 2,652 \text{ tons}$$

(21) which is equivalent to
$$\frac{2,652 \times 12}{44} = 723 \text{ tons of carbon.}$$

- (22) The CO₂ content of the other burden constituents (ore, scale, and cinder) averaged about 1 per cent., giving off an additional

$$(23) \quad \frac{31,372 \times 1}{100} = 314 \text{ tons CO}_2, \text{ which is equivalent to}$$

$$(24) \quad \frac{314 \times 12}{44} = 86 \text{ tons of carbon.}$$

The total amount of carbon which escaped from the furnace during the month in the shape of CO₂ and CO is, therefore (see 18, 21, and 24),

$$(25) \quad 10,537 + 723 + 86 = 11,346 \text{ tons.}$$

Each ton of the top gases containing 0.568 ton (see 2) of carbon, the total weight of the dry top gases for the month is:

$$(26) \quad \frac{11,346}{0.1568} = 72,360 \text{ tons.}$$

The total weight of water, which was charged into the furnace with the ore, coke, and limestone, and which subsequently had to be evaporated, leaving the furnace together with the top gases, was, according to the analysis of the burden constituents,

$$(27) \quad 4,665 \text{ tons.}$$

The nitrogen content of the dry top gases (see 1) originating exclusively from the blast, and the nitrogen content of the air being known, the weight of the dry air which entered into the furnace through the tuyères can be calculated as follows:

$$(28) \quad \frac{72,360 \times 54.9}{77} = 51,592 \text{ tons.}$$

- (29) The average moisture of the atmosphere was 5.49 grains per cubic foot at 70° F., making the weight of natural air blown into the tuyères,

$$(30) \quad 51,592 + 539 = 52,131 \text{ tons.}$$

- (31) The slag volume according to the daily burden calculations averaged 45.5 per cent., making the total amount of slag produced, 7,811 tons.

The weight of materials passed through the furnace was as follows:

Material	Total Tons	Per Ton of Product Including Scrap Produced	
		Tons	Pounds
Burden	31,645	1.827	4,093
Limestone	6,154	0.355	795
Coke	13,006	0.751	1,682
Product (including scrap)	17,319		
(33) Slag	7,811	0.451	1,010
Flue dust	691	0.040	89
Blast (excluding moisture)	51,592	2.979	6,673
Moisture in blast	539	0.031	69
Top gas (excluding moisture)	72,360	4.173	9,359
Moisture in gas	4,665	0.269	603

According to the average monthly analysis the product consisted of,

	C.	Si.	Sul.	Phos.	Mn.	Fe.
(34)	4.20	1.35	0.032	0.071	0.74	93.61 per cent.

The iron content originated from the following amounts of burden constituents:

	Material	Group 1	Group 5	Pewabic Genoa	Pit Cinder	Steel Scale	Scrap	Coke	Total
	Iron, per cent. . . .	52.00	49.00	35.00	50.00	66.00	75.00	0.80	
	Stage of oxidation.	Fe ₂ O ₃	Fe ₂ O ₃	Fe ₂ O ₃	Fe ₂ O ₄	Fe ₂ O ₄	Fe	Fe	
	Pounds per ton of product.	2,819	755	219	58	207	36	1,682	
(35)	Pounds minus 1 per cent. loss. . .	2,791	748	217	57	205	35	1,665	
	Pounds iron per ton of product	1,451	366	76	29	136	26	13	2,097

Consequently the iron content of one ton of product originated from the following stages of oxidation:

	Fe	Fe ₂ O ₄	Fe ₂ O ₃
(36)	39	165	1,893 Pounds

(37) In the same manner it results that one ton of product contained 0.74 per cent. or 16.6 lb. of manganese; 16.3 lb. of which have been reduced from MnO, while 0.3 lb. were contained in the scrap charged.

(38) The phosphorus of the product amounts to 1.6 lb. per ton and has been reduced practically all from P₂O₅.

(39) The 1.35 per cent. of silicon is equivalent to 30.2 lb. per ton, of which 29.7 lb. were reduced from SiO₂, while 0.5 lb. were contained in the scrap charged.

Heat Balance

(a) Heat Generation

The total weight of carbon contained as CO₂ in the top gases per ton of product is (see 2 and 33):

$$(40) \quad 0.0608 \times 9,359 = 569.0 \text{ lb.}$$

The weight of carbon per ton of product, equivalent to the amount of CO₂ originating from the limestone and burden and contained in the top gases is (see 21, 24, and 33):

$$(41) \quad \frac{809 \times 2,240}{17,319} = 104.6 \text{ lb.}$$

Deducting item (41) from (40) gives the weight of coke carbon burned to CO₂ per ton of product: 569.0 - 104.6 = 464.4 lb.

(43) One pound of carbon through combustion to CO₂ generates 14,543 B.t.u.;

(44) the above 464.4 lb. generated, therefore, 6,754,000 B.t.u.

The amount of carbon per ton of product burned to CO is (see 2 and 33):

$$(45) \quad 0.0960 \times 9,359 = 898.5 \text{ lb.}$$

- (46) One pound of carbon through combustion to CO generating 4,446 B.t.u.,
 (47) the above 898.5 lb. generated 3,995,000 B.t.u.
 (48) The specific heat of the air blast being 0.248 and the average hot-blast tem-
 (49) perature 1,243° F., the heat brought into the furnace with the blast (see 33)
 (50) per ton of product is: $1,243 \times 0.248 \times 6,673 = 2,057,000$ B.t.u.

In the same manner the amount of heat brought into the furnace per ton of product with the moisture (specific heat, 0.49) content of the blast (see 33) is:

(51) $1,243 \times 0.49 \times 69 = 42,000$ B.t.u.

(b) *Heat Consumption*

The following data were used in these calculations:

	B.t.u.
Heat required to reduce Fe_2O_3 to 1 lb. of Fe.....	3,240
Heat required to reduce Fe_3O_4 to 1 lb. of Fe.....	2,970
(52) Heat required to reduce MnO to 1 lb. of Mn.....	2,970
Heat required to reduce SiO_2 to 1 lb. of Si.....	14,090
Heat required to reduce P_2O_5 to 1 lb. of P.....	10,620

The reduction of one ton of product therefore requires the following heat (see 34, 36, 37, and 38):

	B.t.u.
1,893 lb. of Fe from Fe_2O_3	6,133,300
165 lb. of Fe from Fe_3O_4	490,000
(53) 16.3 lb. of Mn from MnO.....	48,400
1.6 lb. of P from P_2O_5	17,000
29.7 lb. of Si from SiO_2	418,500
Total.....	7,107,200

The weight of CO_2 produced by the calcination of carbonates per ton of product is, according to (20), (23) and (33):

(54)
$$\frac{(2,652 + 314) \times 2,240}{17,319} = 384 \text{ lb.}$$

practically all of which was contained in the burden in the form of CaCO_3 .

- (55) The heat necessary to decompose CaCO_3 into CaO and CO_2 is 1,830 B.t.u. per pound of CO_2 . The driving off of above 384 lb. required

(56) $1,830 \times 384 = 702,000$ B.t.u.

The amount of heat carried from the furnace by the slag and iron was not determined. The figures used represent the average results of former tests under similar conditions. According to these the heat carried off by 1 lb. of

- (57) iron amounted to 510 B.t.u. and per pound of slag to 900 B.t.u.

The heat carried off per ton of product with the iron amounts to

(58) $510 \times 2,240 = 1,142,500$ B.t.u.

and with the slag

(59) $900 \times 1,010 = 909,000$ B.t.u.

The dissociation of the moisture as carried into the furnace with the blast

- (60) requires, per pound of water vapor, 5,760 B.t.u.; per ton of product, the heat

- (61) required for this reaction was (see 33):

$69 \times 5,760 = 397,000$ B.t.u.

The dry top gases carried off the following amount of heat per ton of product (The average top temperature was 325° F.; (see (1) for the gas analysis and (33) for the weight of gas).

Gas analysis $\times$ Weight of gas $\times$ Top temperature $\times$ Spec. Heat = B.t.u.

	CO ₂	0.223	9,359	325	0.2169	147,100
	CO	0.224	9,359	325	0.2426	165,300
(62)	H ₂	0.003	9,359	325	3.4090	31,100
	CH ₄	0.001	9,359	325	0.5930	1,800
	N ₂	0.549	9,359	325	0.2438	407,100
Total.....						752,400

The moisture carried out by the top gases entered the furnace at a temperature of 70° F., i.e., with a heat content (see 33) per ton of product of

$$(63) \quad (70 - 32) \times 603 = 22,900 \text{ B.t.u.}$$

Since the moisture was heated in the furnace to 212° F. and evaporated, and the resulting steam superheated to 325° F., the heating of the water required

$$(64) \quad (212 - 32) \times 603 - 22,900 = 85,600 \text{ B.t.u.}$$

The evaporation required

$$(65) \quad 964.8 \times 603 = 581,800 \text{ B.t.u.}$$

The superheating required

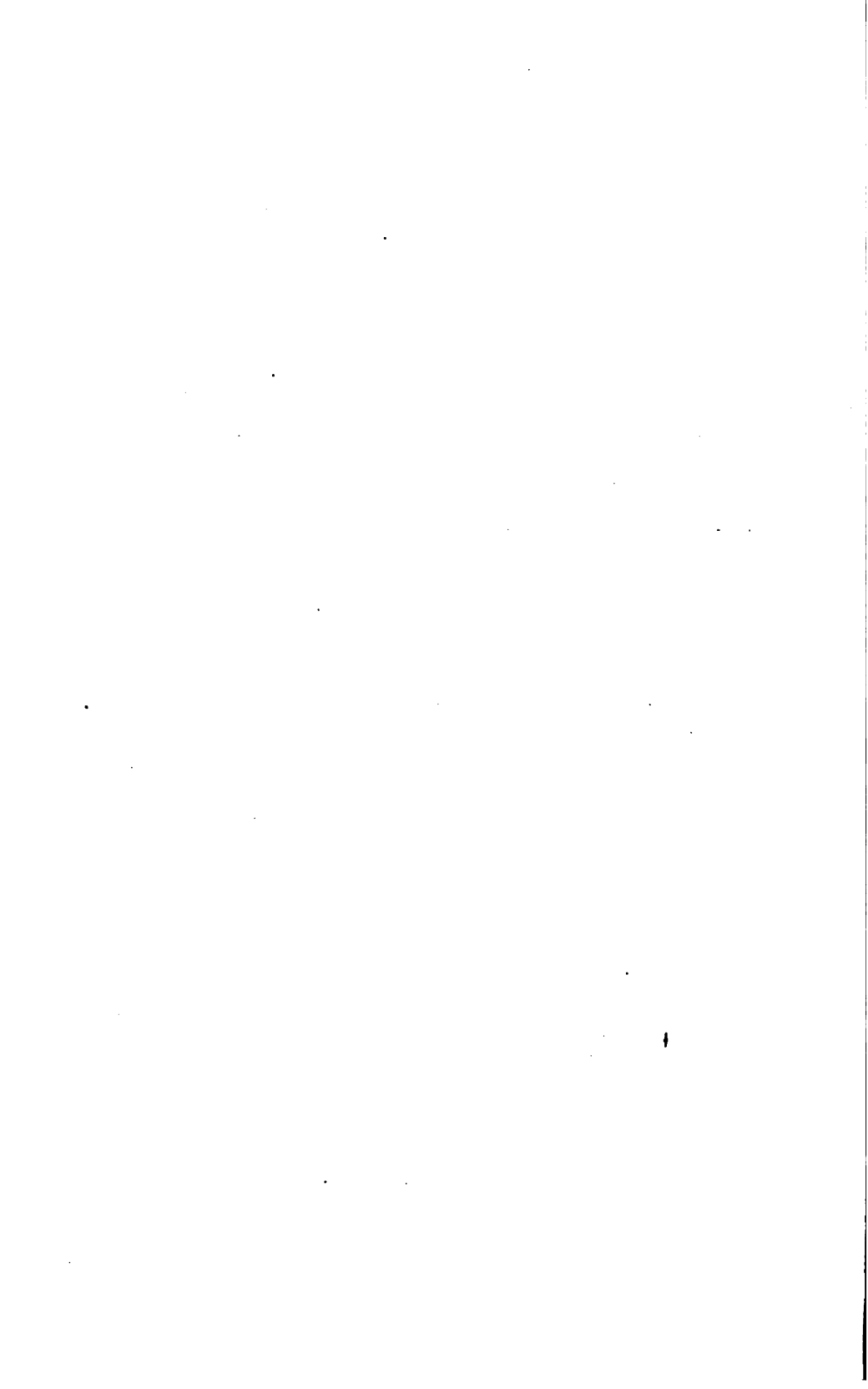
$$(66) \quad 0.48 \times (325 - 212) \times 603 = 32,700 \text{ B.t.u.}$$

Therefore the total heat carried off with the moisture of the top gases per ton of product amounts to

	B.t.u.	
	85,600	(see 64)
	581,800	(" 65)
	32,700	(" 66)
(67)	<u>700,100</u>	

Heat Balance Sheet for One Ton of Product

Generated			Consumed		
By	B.t.u.	Per Cent.	By	B.t.u.	Per Cent.
Combustion C to CO	3,995,000	31.1	Reduction of Fe ₂ O ₃	6,133,300	
Combustion C to CO ₂	6,754,000	52.6	Reduction of Fe ₂ O ₃	490,000	6,623,300
Hot blast (heat content)	2,057,000	16.0	Reduction of MnO		48,400
Moisture of the air in hot blast (heat content)	42,000	0.3	Reduction of P ₂ O ₅	17,000	
			Reduction of SiO ₂	418,500	
			Calcination of carbonates	702,000	5.5
			Dissociation of moisture in blast	397,000	3.1
			Carried off with the iron	1,142,500	8.9
			Carried off with the slag	909,000	7.1
			Carried off with the dry top gases	752,400	5.9
			Carried off with the moisture in top gas	700,100	5.4
			Radiation, cooling water, and unaccounted for	1,137,800	8.8
Total	12,848,000	100.0	Total	12,848,000	100.0



DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Effect of Finishing Temperatures of Rails on Their Physical Properties and Microstructure

BY W. R. SHIMER, SO. BETHLEHEM, PA.

(New York Meeting, February, 1915)

IN his valuable report on Finishing Temperatures and Properties of Rails,¹ Dr. G. K. Burgess, Chief of the Division of Metallurgy, U. S. Bureau of Standards, has begun a line of investigation which should be continued by those interested in the subject, and who have proper facilities for carrying out the work. For the past year or more the Bethlehem Steel Co. has been conducting experiments to learn the effect of rail finishing temperatures on their physical properties and microstructure and the results are here given for the consideration of those interested in the rail situation.

Considerable differences of opinion have been expressed by various authorities, in the past, as to the effect of large or small grain size and high or low finishing temperatures of steel rails on their physical properties and wearing qualities. Low finishing temperatures and, therefore, low shrinkage, have been advocated on the theory that the wearing qualities would be improved. Others have recommended high finishing temperatures.

The generally accepted cause for rail failures, due to the development of transverse fissures, is excessive wheel pressure on hard rails. One of the causes advanced for hard rails has been that they contain too high percentages of carbon and manganese; another that the rails were finished at too low a temperature, *i.e.*, at or near the critical point.

In the experiments herein described the rails were rolled from reheated blooms. All the rail blooms were charged hot in a reheating furnace and brought up to about the original ingot-rolling temperature before rolling into rails. These rails gave better results in deflection, withstood a greater number of drops before breaking, and showed greater ductility, than rails of identically the same composition and section which were rolled direct from the ingot. Rails rolled from reheated blooms, finished at somewhat higher temperatures than when rolled direct from the ingot, have a different microstructure, which the writer does not believe is due so much to the difference in finishing temperatures, as to the difference in

¹*Technologic Paper No. 38, N. S. Bureau of Standards (1914); Bulletin No. 93, Sept., 1914, pp. 2433 to 2437.*

Heat C.—100-lb. Rails Rolled Direct (Compare with Heat D)

C, 0.646; Mn, 0.81; Si, 0.105; P, 0.028; S, 0.029

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
2	1	1.30	0.03	0.03	0.05	0.05	0.05	0.03	0.24
	2	2.30	0.04	0.06	0.08	0.09	0.09	0.08	0.44
	3	3.30	0.07	0.09	0.13	0.15	0.14	0.11	0.69
Middle	1	1.20	0.02	0.03	0.04	0.05	0.04	0.04	0.22
	2	2.30	0.06	0.07	0.08	0.09	0.08	0.07	0.45
Last	1	1.20	0.03	0.04	0.05	0.05	0.04	0.03	0.24

Average deflection after the first blow, 1.23 in.

Heat D.—100-lb. Rails Rolled from Reheated Blooms

C, 0.648; Mn, 0.83; Si, 0.081; P, 0.028; S, 0.031

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
1	1	1.45	0.04	0.04	0.06	0.05	0.04	0.03	0.26
	2	2.60	0.06	0.08	0.12	0.09	0.09	0.05	0.49
	3	3.70	0.10	0.13	0.17	0.15	0.12	0.09	0.76
	4	4.70	0.10	0.13	0.17	0.17	0.15	0.11	0.83
	5	Broke	0.12	0.14	0.18	0.18	0.15	0.11	0.88
2	1	1.45	0.06	0.07	0.07	0.06	0.04	0.03	0.33
3	1	1.50	0.04	0.05	0.06	0.05	0.04	0.03	0.27

Average deflection after the first drop, 1.47 in.

Heat E.—100-lb. Rails Rolled Direct (Compare with Heat F)

C, 0.740; Mn, 0.78; Si, 0.104; P, 0.020; S, 0.033

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
2	1	1.01	0.04	0.05	0.03	0.03	0.02	0.02	0.19
	2	1.90	0.05	0.06	0.07	0.08	0.05	0.05	0.36
10	1	1.01	0.04	0.04	0.04	0.04	0.02	0.02	0.20
19	1	1.01	0.03	0.04	0.05	0.05	0.04	0.03	0.24

Average deflection after the first drop, 1.01 in.

Heat F.—100-lb. Rails Rolled from Reheated Blooms

C, 0.750; Mn, 0.81; Si, 0.071; P, 0.018; S, 0.029

Ingot No.	No. Drops	Permanent Set, Inches	Elongation						
			1 in.	2 in.	3 in.	4 in.	5 in.	6 in.	Total
1	1	1.50	0.04	0.05	0.05	0.05	0.03	0.03	0.25
	2	2.80	0.06	0.09	0.11	0.10	0.09	0.07	0.52
	3	3.70	0.09	0.12	0.15	0.15	0.14	0.09	0.74
	4	4.80	0.10	0.15	0.18	0.18	0.15	0.10	0.86
	5	6.00	0.12	0.16	0.19	0.19	0.16	0.12	0.94
2	Not tested to destruction								
	1	1.40	0.06	0.06	0.07	0.07	0.05	0.04	0.35
3	1	1.30	0.02	0.04	0.05	0.05	0.04	0.03	0.23

Average deflection after the first drop, 1.40 in.

The drop-test results of the rails rolled from reheated blooms were superior to those of rails rolled direct from the ingot.

Numerous records of direct-rolled rails show the results of drop tests from the same heat to be not uniform with test pieces of uniform chemical composition, taken from the same position in the ingot and free from segregation.

Drop tests of rails rolled from reheated blooms show consistently uniform results under the same conditions.

The reason rails rolled direct do not show uniform drop-test results seems to be that, when two or three blooms are rolled from an ingot, rails rolled from the first bloom are finished at the highest temperature, rails rolled from the second bloom are finished at the next highest temperature, while the rails rolled from the last bloom of the ingot are finished at a considerably lower temperature than the first two. If there are any delays at the mill and any blooms are held up they sometimes become too cold to roll, and if they are just hot enough to roll the rails will be finished below the average temperature.

In the case of rails rolled from reheated blooms a uniform temperature can be obtained on all rails rolled. One bloom is drawn from the reheater at a time and sent through the mill. If there is any delay at the mill the blooms remain in the reheater until the mill is ready to roll them. Any desired temperature can be maintained in the reheating furnace so as to obtain the desired finishing temperature on the rails. The temperature of the reheater is regulated according to the composition and section of the rails being rolled.

Later in this report, results of an investigation on finishing rails at different temperatures are recorded. Nine 19 by 23 in. ingots from the same heat were each rolled into two "three-rail" blooms, each ingot mak-

ing two blooms, or six rails 33 ft. long of 100-lb. section. The composition of this heat was: C, 0.720; Mn, 0.73; P, 0.019; S, 0.037. These 18 blooms were charged direct from the bloomer into the reheating furnace and heated up to the original ingot-rolling temperature and, after drawing from the furnace, some were held for different lengths of time, before rolling, so as to obtain a range of high and low finishing temperatures on the rails rolled from them.

Finishing temperatures were measured on all the rails directly after leaving the finishing rolls. Shrinkages were measured on all the rails. A drop-test piece, 4 ft. long, was cut from each *A*, *B*, *C*, *D*, *E* and *F* rail from each of the nine ingots, making a total of 54 drop-test pieces.

A short piece, 6 in. long, was cut from each rail directly back from where the drop-test pieces were cut. A standard tensile bar (0.505 in. diameter by 2 in. long) was turned up from both sides of the head of these short rail sections—one from the bottom side of the head of each section as it cooled on the bed, and one from the top side—making 108 tensile bars.

Next to the piece for tensile test a thin section was sawed from the head of each of the 54 rails, and used for making Brinell hardness tests.

Finishing temperatures ranging from 2,147° to 1,652° F. were obtained in this experiment.

For convenient study, the results of drop tests, physical tests, Brinell tests and check carbons on all the rails are given in the accompanying table. In the table are given the minimum deflection, or the deflection after the first blow of the drop; the maximum deflection, or the last deflection measured before the drop-test piece broke; the number of drops of the tup required to break the rail; the total elongation measured in 6 in., after the rail broke; the tensile results of the bar cut from the top side of the head of the rail as it cooled on the bed; the tensile results of the bar from the bottom side of the head as it cooled on the bed.

Ingots were numbered from one to nine, inclusive. The positions of the rails in each ingot were as follows: *A* the top rail of the ingot, *B* the second from the top, and so on down, *F* being the bottom rail of the ingot. The *A*, *B*, and *C* rails of all ingots were rolled from the upper bloom; the *D*, *E*, and *F* rails were rolled from the bottom bloom of the ingots.

The shrinkages recorded are averaged from measurements of the three rails rolled from each of the 18 blooms.

The finishing temperatures are fairly consistent with the shrinkages, but there are a few cases where they are not quite so, due to the difficulty experienced by the operator of the optical pyrometer in matching the color correctly.

A microscopic examination was made of the threaded end of each of the 108 tensile bars. The micrographs shown at the end of this report reproduce the average structure of the top and bottom tensile bars from

the 6-in. rail pieces from the nine ingots. All micrographs are magnified 100 diameters.

Drop Tests.—All rails were tested head up. Six 1-in. spaces were marked with a center punch on the base of the test piece, and elongations were measured from these punch marks after the rail broke. In some cases the rails bent almost at right angles without breaking in the base; in these cases the elongations are not as great as they would have been had the rail broken in the base. Figs. 10 and 11 show some of the test pieces which bent to almost 90° without breaking in the base, proving them to be exceedingly ductile, although the ductility measurement does not indicate this. The height of drop was 18 ft. and the weight of the tup 2,000 lb.

The following summary is drawn from a practical standpoint based on the results of the various tests and microscopic examinations and a study of numerous records.

Summary

No appreciable difference in grain was found to indicate that the size or structure was governed by a difference in finishing temperature.

There was practically no free ferrite in any of the *A* rails, slightly more in the *B* rails, more in the *C* rails, and so on; the *E* and *F* rails from all the nine ingots contained an almost complete network of free ferrite.

The *A*, *B*, *C*, and *D* rails all had a more or less sorbitic structure, and the *E* and *F* rails were not sorbitic. The only explanation the writer can offer for this consistent difference between the microstructures of the various rails according to their position in the ingot is as follows:

(1) The absence of free ferrite in the *A*, *B*, and *C* rails seems to be due partly to the presence of a consistently higher percentage of carbon than in the *D*, *E*, and *F* rails.

(2) When these experimental rails were rolled, the rails from each ingot were kept apart on the bed to avoid their being mixed. For example: When Bloom 1*ABC* was rolled, the *A* rail was sawed to length first, and the *C* rail last. When they were put on the bed, the *C* rail was first, the *B* rail next with its base resting against the head of the *C* rail, and the *A* rail last with its base against the head of the *B* rail. These rails were cooling at least 3 to 5 min. before the *D*, *E*, and *F* rails from the bottom bloom of No. 1 ingot came along. When the *D*, *E*, and *F* rails went over to the bed, the *F* rail went first, a small space being left between it and the *A* rail; the base of the *E* rail rested against the head of the *F* rail, and the base of the *D* rail rested against the head of the *E* rail. The head of the *D* rail was left exposed. Therefore, the rails from the various ingots cooled at different rates of speed. The heads of all the *A* rails were air cooled and the structure was sorbitic. The heads of the *B* and *C* rails cooled more slowly on account of having hot rails resting against them, consequently these rails contained more free ferrite than the *A* rails. The

C rails contained more ferrite than the *B*, and the *B* more than the *A* rails. The head of the *C* rails had an annealing effect longest, the *B* next, and the *A* rail none.

Now, as to the *D*, *E*, and *F* rails. The *F* rail was the first to have a hot rail resting against its head, the *E* rail next, while the head of the *D* rail was air cooled. This, it seems to me, shows that this consistent difference in the separation of free ferrite, according to the position of these rails in the ingot, is due simply to the difference in the rate of cooling of the rails.

In order to learn what would happen if samples of *A*, *B*, *C*, *D*, *E*, and *F* rails from the same ingot were all annealed at the same temperature and cooled at the same rate of speed, samples from rails 1*A*, 1*B*, 1*C*, 1*D*, 1*E*, 1*F*, 9*A*, 9*B*, 9*C*, 9*D*, and 9*E* were charged in the same muffle furnace, heated to slightly above the critical point (1,450° F.), and left to cool slowly in the furnace over night. Micrographs were taken of the structures of these pieces after annealing and are shown in Figs. 12 and 13.

It will be seen that after this annealing, the structures of all the rails, except 1*F*, were about the same, and contained no free ferrite. Rail 1*F* contained a network of free ferrite, but as this sample contained only 0.622 per cent. carbon and the other 10 samples contained no less than 0.680 per cent., the difference in structure of this sample could be due to its lower carbon content. In the other 10 samples the aggregate of ferrite and pearlite has changed to a homogeneous solid solution.

The difference in carbon, together with the difference in the rate of cooling, accounts for the separation of free ferrite in some instances and the non-separation in others.

The above samples were afterward heated to 1,800° F. and allowed to cool slowly in the furnace over night. These samples were polished and etched and micrographs taken of their respective structures. In this case the grain size increased and free ferrite separated out in the boundaries of the pearlite crystals. Approximately the same amount of free ferrite was observed in all the pieces, showing that when heated to a rail-finishing temperature and allowed to cool at the same rate of speed the microstructure is the same throughout, regardless of the position of the rail in the ingot. These micrographs are shown in Figs. 14 and 15.

The average physical results of the 54 tensile bars tested from all the *A*, *B*, and *C* rails are as follows:

Tensile Strength	Elastic Limit	Elongation, Per Cent.	Contraction, Per Cent.
125,130	64,150	14.0	19.12

The average physical results of the 54 tensile bars tested from all the *D*, *E*, and *F* rails are as follows:

Tensile Strength	Elastic Limit	Elongation, Per Cent.	Contraction, Per Cent.
116,300	59,720	16.2	24.54

The average carbon content of the *A*, *B*, and *C* rails is 0.711 per cent.; of the *D*, *E*, and *F* rails, 0.678 per cent.

The *A*, *B*, and *C* rails are three points higher in carbon; higher in tensile strength; higher in elastic limit; and lower in elongation and contraction of area than the *D*, *E*, and *F* rails. Apparently the slight difference in carbon content has caused the difference in physical properties. Check analyses were made for manganese, silicon, phosphorus, and sulphur, and all the results were so close to the original heat analysis that the difference in physical properties cannot be due to these elements.

The following are the average carbon contents and average tensile results (each an average of six tests), finishing temperatures, and shrinkage of the rails rolled from each bloom:

ABC Rails, Ingot	Tensile Strength	Elastic Limit	Elonga- tion, Per Cent.	Contraction, Per Cent.	Carbon, Per Cent.	Finishing Temperature, °F.	Shrinkage, Inches
1	128,000	66,600	14.6	17.91	0.712	2,146	6.60
2	124,330	64,660	12.8	16.90	0.713	2,138	6.69
3	125,910	64,330	13.9	20.20	0.702	2,049	6.64
4	128,000	64,660	14.2	19.04	0.711	2,120	6.77
5	125,660	65,180	11.9	17.39	0.700	2,147	6.64
6	123,660	63,110	14.4	20.56	0.700	2,012	6.37
7	124,500	63,350	14.1	19.84	0.706	1,868	6.06
8	124,500	63,330	16.0	20.30	0.733	1,850	5.85
9	121,600	61,800	14.1	19.98	0.714	1,832	5.66
Average....	125,130	64,150	14.0	19.12	0.711		
DEF Rails, Ingot							
1	118,750	60,500	15.6	24.71	0.669	2,020	6.44
2	116,580	59,660	16.5	23.90	0.664	2,084	6.63
3	116,830	60,670	15.9	23.07	0.667	2,120	6.71
4	116,100	59,500	15.7	22.41	0.650	2,147	6.65
5	116,000	59,500	16.1	24.09	0.676	2,048	6.56
6	116,100	60,400	15.7	22.07	0.665	1,904	6.37
7	115,660	59,500	16.6	25.81	0.674	1,850	5.81
8	115,400	59,250	16.9	24.83	0.681	1,841	5.69
9	115,330	58,910	17.0	30.00	0.680	1,652	5.37
Average....	116,300	59,720	16.2	24.54	0.678		

The above shows that the rails finished at high temperatures have the same strength and ductility as those finished at low temperatures, this conclusion being confirmed by a study of the drop-test results and the micrographs. The latter show that the structure of all the *A* rails is identical; all the *B* rails have the same structure; in short, all rails from the same relative position in the various ingots show the same micro-

structure. The only difference found, regardless of finishing temperatures, was between the physical properties of the *A*, *B*, and *C* rails and the *D*, *E*, and *F* rails, which was dependent upon the position of the rails in the ingot. The difference in microstructure was due to the rate of cooling when the rails were on the bed.

The above results refer to rails rolled from reheated blooms. From past experience it has been noticed that the rails rolled direct from the ingot do show differences in physical properties, varying in accordance with their finishing temperatures. The difference, it seems, is due to the fact that when rolling direct from a 19 by 23 in. ingot, or an 18 by 19 in. ingot, considerable strains are set up during this continued reduction, which are increased or decreased according to the finishing temperature.

When rails are rolled from reheated blooms, all the strains set up during the reduction of the ingot to the bloom are removed during the reheating process, since the blooms are reheated to or near the original ingot-rolling temperature. The percentage of reduction from the bloom to the finished rail is not so great as from the ingot to the finished rail, consequently no such strains are set up and therefore a difference in finishing temperature does not seem to affect appreciably the physical properties and microstructure of rails rolled from reheated blooms.

A careful study of the results of the drop tests recorded in this report shows that the rails finished at high temperatures are equally as good as those finished at the lower temperatures. Figs. 10 and 11 show drop-test pieces representing the following rails:

1-E—Finished at 2,020° F.; shrinkage 6.44 in.; stood six drops and did not break in the base.

1-F—Finished at 2,020° F.; shrinkage 6.44 in.; stood six drops and did not break in the base.

3-B—Finished at 2,048° F.; shrinkage 6.64 in.; stood five drops and did not break in the base.

4-C—Finished at 2,120° F.; shrinkage 6.77 in.; stood five drops and did not break in the base.

5-D—Finished at 2,048° F.; shrinkage 6.55 in.; stood five drops and did not break in the base.

9-B—Finished at 1,832° F.; shrinkage 5.66 in.; stood six drops and did not break in the base.

In addition to those shown in the photographs, 9-D and 9-E finished at 1,652° F.; shrinkage 5.37 in.; stood six drops and five drops respectively and broke in the same manner as those illustrated.

The following drop-test pieces also broke in the manner above described: 1-A, 1-D, 1-E, 1-F, 2-A, 3-A, 3-B, 3-C, 3-D, 4-C, 4-D, 4-E, 5-A, 6-A, 6-B, 6-C, 7-D, 9-B, 9-D and 9-E. These drop-test pieces represent a wide range of finishing temperatures, shrinkage, and microstructure,

but have almost identical physical characteristics as shown by the results of this test, which is the important test of the quality of a rail.

The conclusions above are based on the results of a carefully carried out investigation along practical lines. The primary object of these experiments was to determine the best finishing temperature for steel rails. There was a variation of 500° F. in the finishing temperatures and a variation of 1.34 in. in shrinkage. Even with this wide range in temperature and shrinkage, it was impossible to determine, from the results of the respective tests, at what temperature a rail should be finished in order to obtain the best results, since all the tensile and drop-test results recorded are good, and practically identical. The results show it is equally impossible to decide what microstructure and shrinkage are desirable.

Further, from the results of this investigation we have learned that rails rolled from reheated blooms are more ductile than those rolled direct from the ingot, regardless of their finishing temperature. Also, we have learned to be cautious with regard to judging from the microstructure as to the temperature at which a rail was finished and to be careful in drawing conclusions from the microstructure, alone, in rail-failure investigations. The microstructure appears to be controlled more by the rate of cooling from above the critical point than it does by the temperature at which it was finished. In regular practice rails cool uniformly, since hot rails are going to the bed continually, and therefore the microstructures do not vary as much as they have in this experiment.

Further investigations on this subject are being conducted and the results may be published at some future date.

[illegible]

EFFECT OF FINISHING TEMPERATURES OF RAILS

Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 in.	Brinell Hardness	Carbon Per Cent.	Tensile Strength	Elastic Limit	Elongation in 2 in.	Contraction of Area
3-A	1.10	3.80	5	0.69	238	0.704	{ Top Bottom	65,000	13.5	19.88
3-B	1.10	3.95	5	0.60	224	0.704	{ Top Bottom	65,000	13.0	20.59
3-C	1.10	3.75	5	0.90	234	0.700	{ Top Bottom	66,000	12.5	15.56
3-D	1.15	5.00	6	0.79	226	0.690	{ Top Bottom	68,000	13.0	17.02
3-E	1.20	4.10	5	0.87	241	0.664	{ Top Bottom	63,000	15.5	24.08
3-F	1.15	3.15	4	0.92	234	0.648	{ Top Bottom	62,000	16.0	24.08
Averages: 3-A, 3-B, 3-C: Fin. Temp. 2,048° F., Shrinkage 6.94 in.; 3-D, 3-E, 3-F: Fin. Temp. 2,120° F., Shrinkage 6.71 in.										
4-A	1.10	3.80	5	0.72	222	0.720	{ Top Bottom	128,000	13.0	18.8
4-B	1.15	3.80	5	0.86	231	0.704	{ Top Bottom	128,000	12.5	18.8
4-C	1.10	4.00	5	0.60	235	0.710	{ Top Bottom	129,000	18.5	18.8
4-D	1.10	4.05	5	0.68	243	0.686	{ Top Bottom	130,000	12.5	17.02
4-E	1.20	5.75	6	0.63	221	0.640	{ Top Bottom	125,000	13.5	21.29
4-F	1.25	4.15	5	0.91	227	0.624	{ Top Bottom	122,000	15.0	19.52
Averages: 4-A, 4-B, 4-C: Fin. Temp. 2,120° F., Shrinkage 6.77 in.; 4-D, 4-E, 4-F: Fin. Temp. 2,147° F., Shrinkage 6.65 in.										
							{ Top Bottom	64,000	14.0	18.8
							{ Top Bottom	63,000	14.0	17.73
							{ Top Bottom	60,000	15.0	20.59
							{ Top Bottom	60,000	15.5	24.08
							{ Top Bottom	50,000	17.5	27.5
							{ Top Bottom	59,000	18.0	25.79

Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 in.	Brinell Hardness	Carbon Per Cent.	Tensile Strength	Elastic Limit	Elongation in 2 in.	Contraction of Area
5-A	1.10	4.90	6	0.80			{ Top Bottom	66,000	13.5	17.73
5-B	1.10	3.80	5	0.83	236	0.700	{ Top Bottom	68,000	7.0	8.90
5-C	1.15	3.85	5	0.95	239	0.700	{ Top Bottom	67,000	12.0	17.02
5-D	1.15	5.20	6	0.81	234	0.688	{ Top Bottom	66,000	12.5	18.8
5-E	1.20	4.10	5	0.98	231	0.654	{ Top Bottom	61,000	14.0	20.59
5-F	1.25	4.25	5	0.97	237		{ Top Bottom	63,000	15.0	21.29
								62,000	15.5	19.52
								60,000	14.0	21.98
								60,000	16.0	23.03
								56,000	15.0	23.03
								57,000	18.0	29.16
									18.5	27.83
Averages: 5-A, 5-B, 5-C: Fin. Temp. 2,147° F., Shrinkage 6.64 in.; 5-D, 5-E, 5-F: Fin. Temp. 2,048° F., Shrinkage 6.56 in.										
6-A	1.10	3.90	6	0.59	233	0.700	{ Top Bottom	60,000	16.0	22.33
6-B	1.10	3.95	5	0.74	237		{ Top Bottom	61,000	13.5	18.8
6-C	1.15	4.10	5	0.64	234	0.700	{ Top Bottom	68,000	12.0	17.02
6-D	1.15	3.00	4	0.78	238	0.710	{ Top Bottom	68,000	14.0	18.8
6-E	1.25	3.20	4	0.83	231	0.650	{ Top Bottom	62,500	16.0	24.08
6-F	1.20	5.30	7	0.99	233	0.636	{ Top Bottom	61,000	15.0	22.33
								63,000	13.5	20.59
								63,000	14.0	17.02
								58,000	15.5	18.8
								59,000	16.0	20.93
								59,000	17.5	26.47
								59,000	18.0	27.5

Averages: 6-A, 6-B, 6-C: Fin. Temp. 2,012° F., Shrinkage 6.37 in.; 6-D, 6-E, 6-F: Fin. Temp. 1,904° F., Shrinkage, 6.37 in.

Rail No.	Minimum Deflection	Maximum Deflection	No. of Drops to Break	Total Elongation in 6 In.	Brinell Hardness	Carbon Per Cent.	Tensile Strength	Elastic Limit	Elongation in 2 In.	Contraction of Area
9-A	1.10	3.70	5	0.99	233	0.718	{ Top 123,000 Bottom 122,500	64,000	12.5	18.8
9-B	1.15	4.10	6	0.65	233	0.716	{ Top 126,000 Bottom 124,000	62,500 65,000	13.0 13.5	18.8 18.8
9-C	1.20	4.05	6	0.62	237	0.708	{ Top 119,000 Bottom 116,000	61,000 60,000	15.0 13.5	21.19 18.8
9-D	1.15	5.35	6	0.61	232	0.680	{ Top 119,000 Bottom 120,000	59,000 60,000	16.0 17.0	22.33 27.16
9-E	1.20	4.15	5	0.79	221	0.680	{ Top 118,000 Bottom 113,000	60,000 59,000	15.5 17.0	23.03 24.08
9-F	1.25	4.25	5	0.90	237		{ Top 112,000 Bottom 110,000	57,000 56,000	18.5 19.0	30.16 30.83

Averages: 9-A, 9-B, 9-C: Fin. Temp. 1,832° F., Shrinkage 5.66 in.; 9-D, 9-E, 9-F: Fin. Temp. 1,652° F., Shrinkage, 5.37 in.

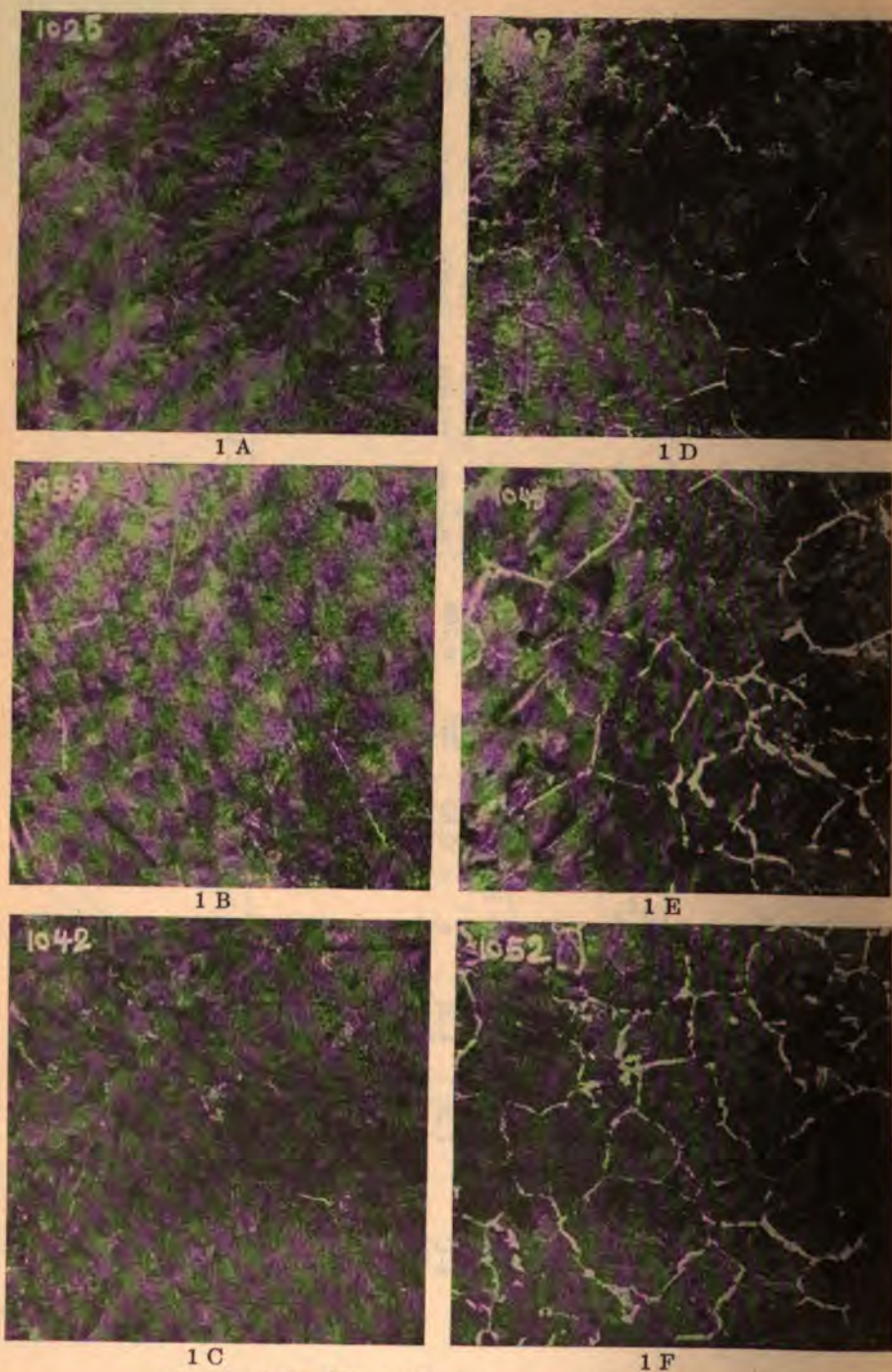
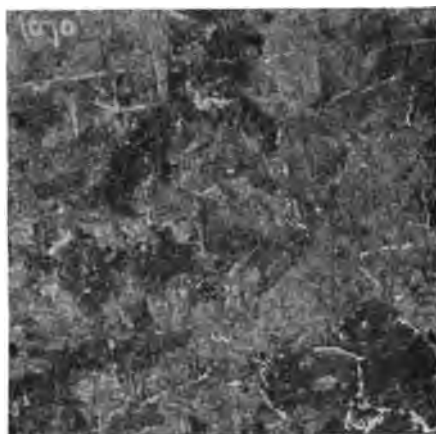


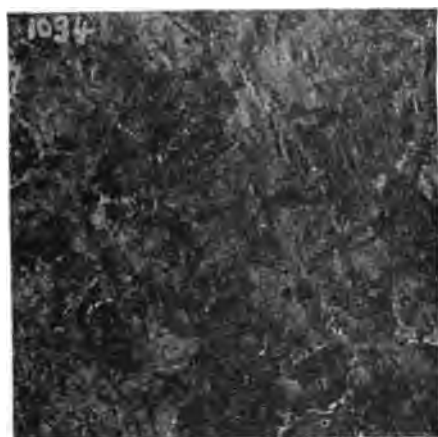
FIG. 1.—INGOT NO. 1. SEE TABULATED DATA, p. 11.



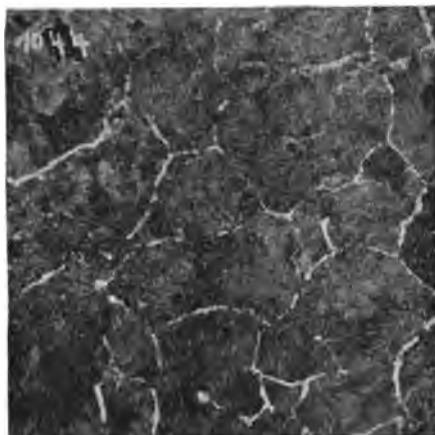
2 A



2 D



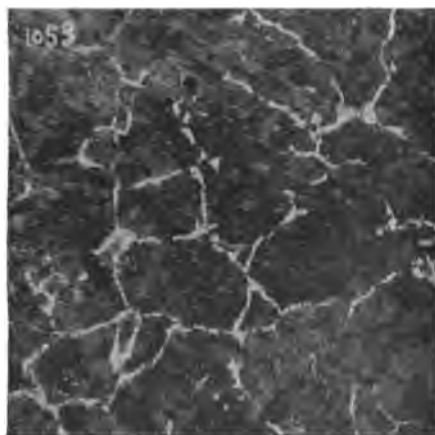
2 B



2 E



2 C

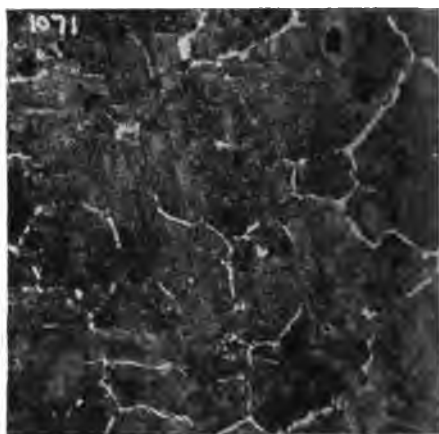


2 F

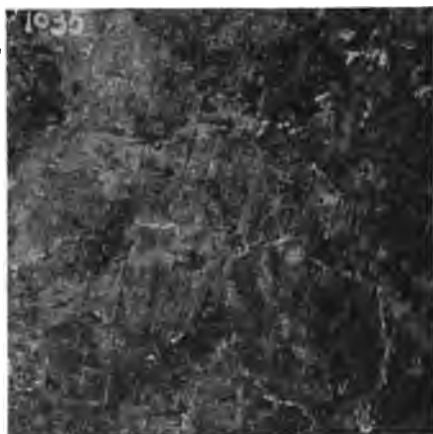
FIG. 2.—INGOT No. 2. SEE TABULATED DATA, p. 11.



3 A



3 D



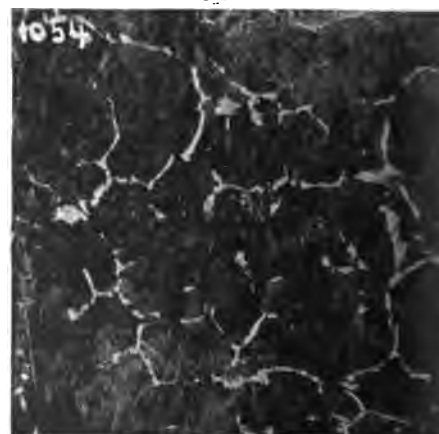
3 B



3 E

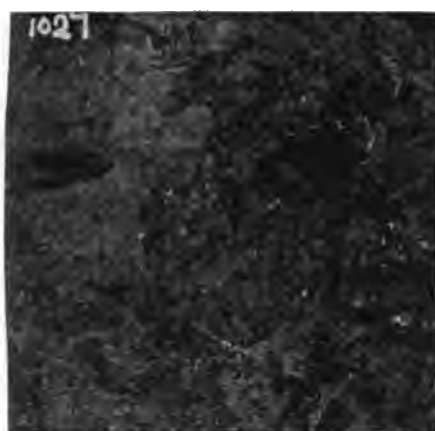


3 C

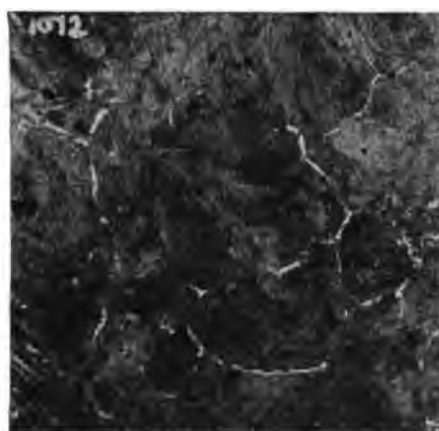


3 F

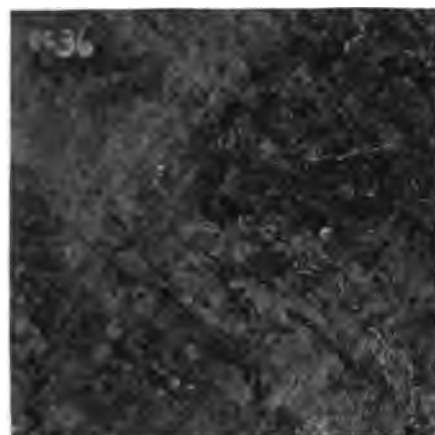
FIG. 3.—INGOT No. 3. SEE TABULATED DATA, p. 12.



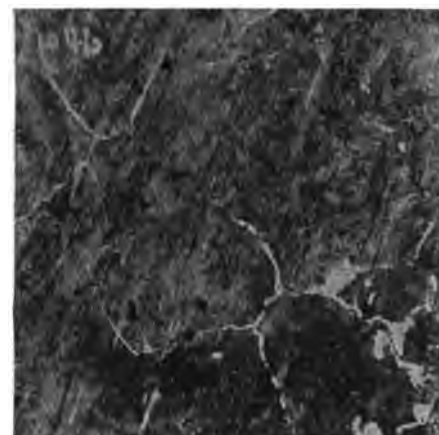
4 A



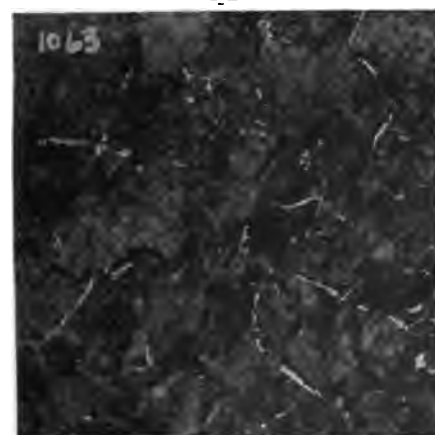
4 D



4 B



4 E

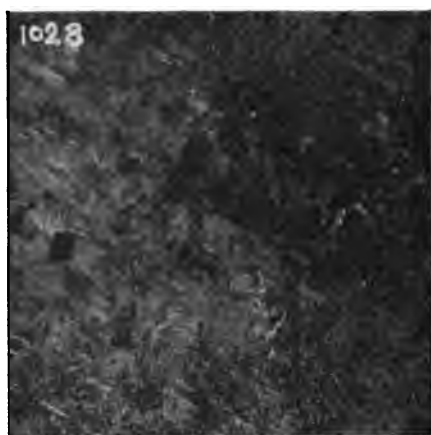


4 C

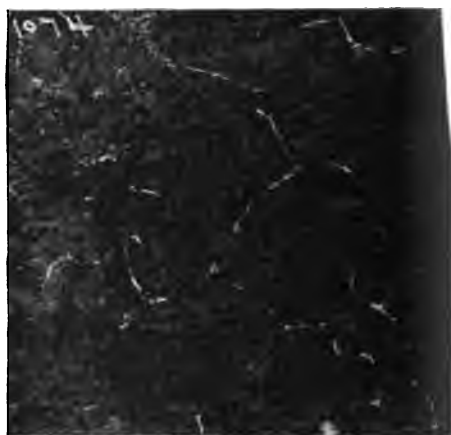


4 F

FIG. 4.—INGOT No. 4. SEE TABULATED DATA, p. 12.



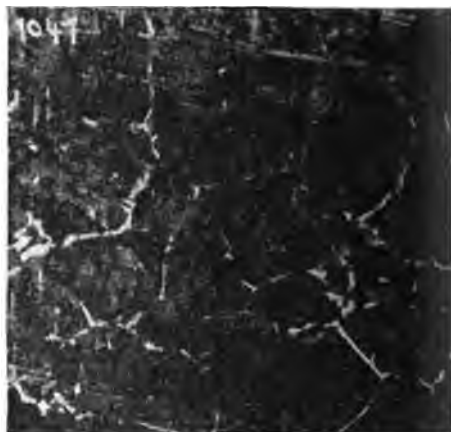
5 A



5 D



5 B



5 E



5 C



5 F

FIG. 5.—INGOT NO. 5. SEE TABULATED DATA, p. 13.



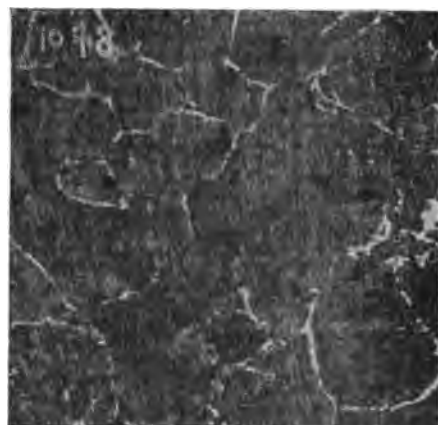
6 A



6 D



6 B



6 E



6 C



6 F

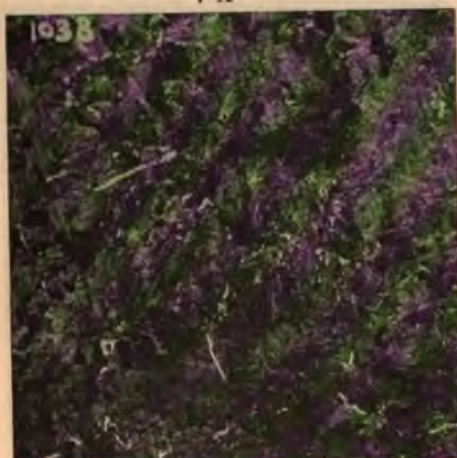
FIG. 6.—INGOT No. 6. SEE TABULATED DATA, p. 13.



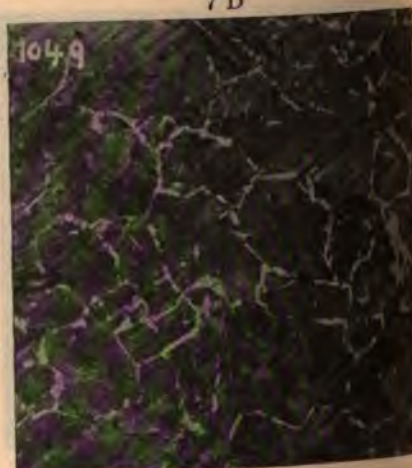
7 A



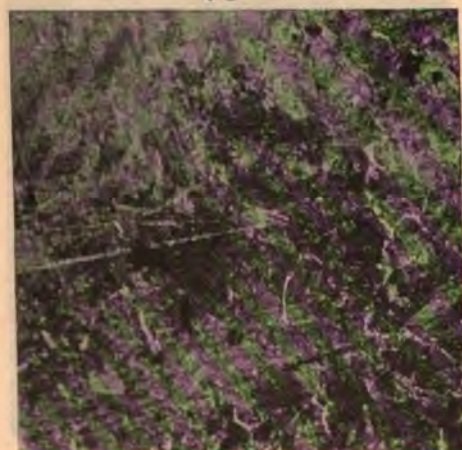
7 D



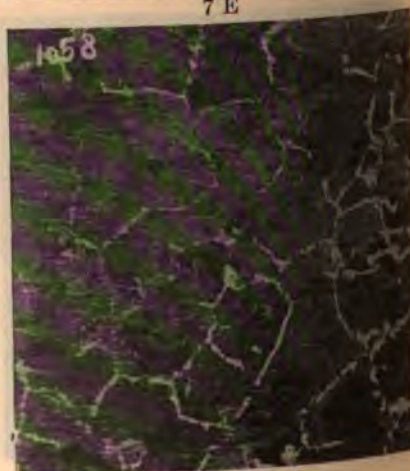
7 B



7 E



7 C



7 F

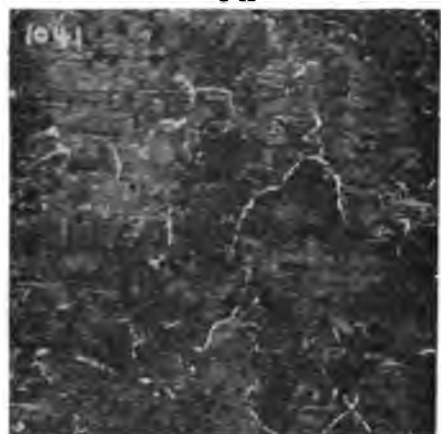
FIG. 7.—INGOT NO. 7. SEE TABULATED DATA, p. 14.



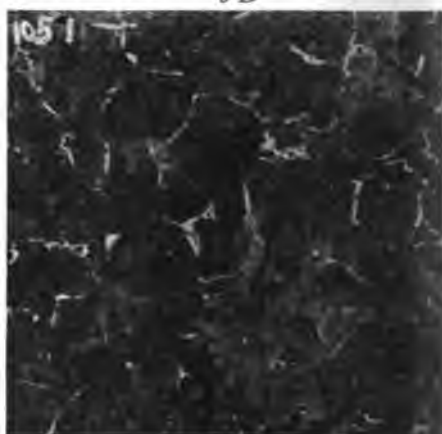
9 A



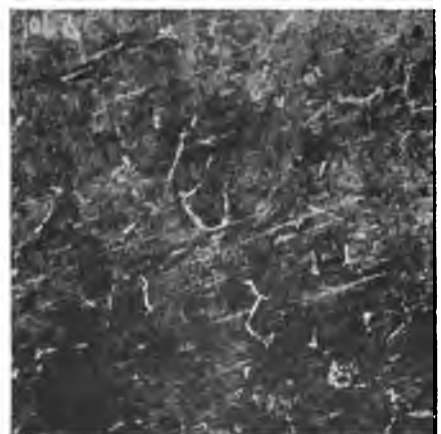
9 D



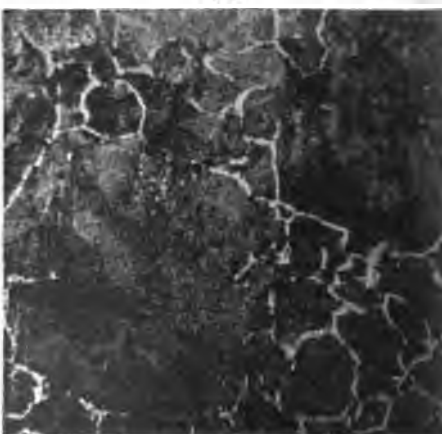
9 B



9 E



9 C



9 F

FIG. 9.—INGOT NO. 9. SEE TABULATED DATA, p. 15.

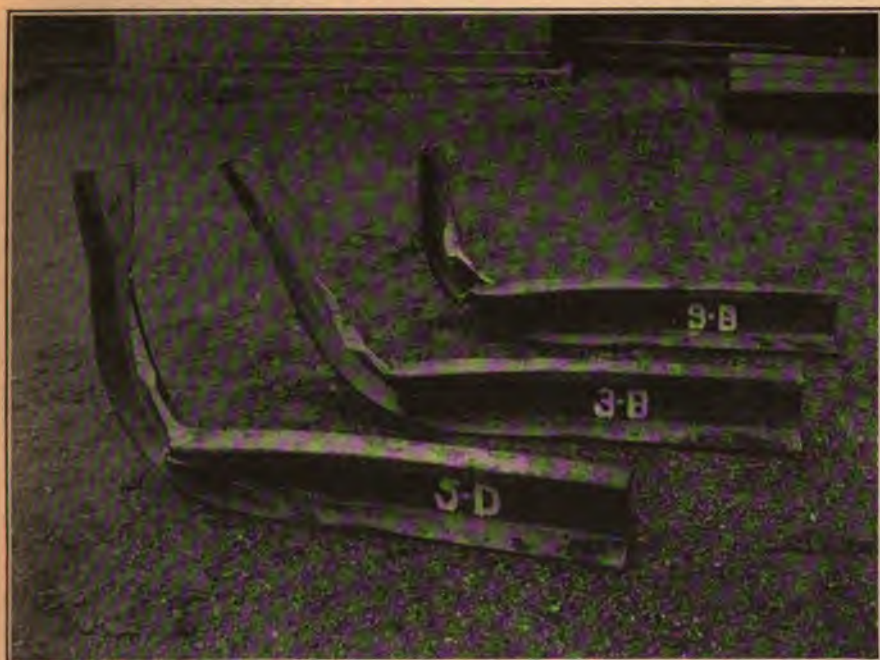


FIG. 10.—SOME OF THE RAILS AFTER BEING SUBJECTED TO DROP TESTS.

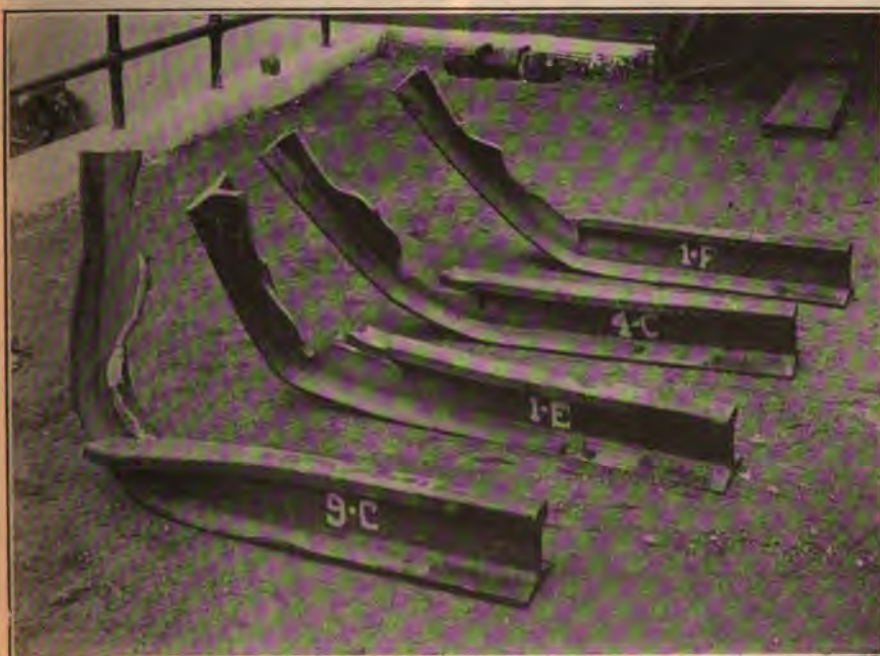
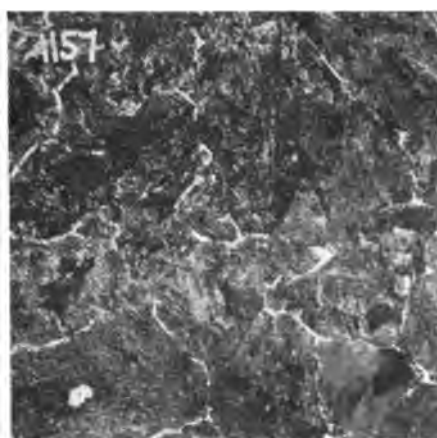
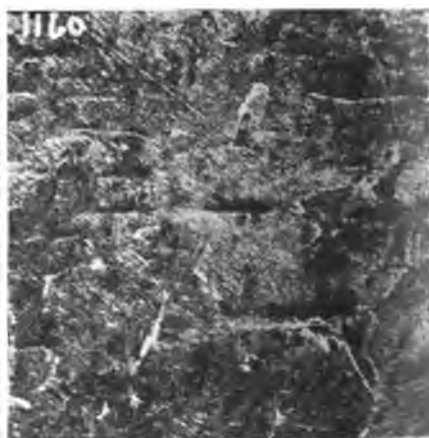


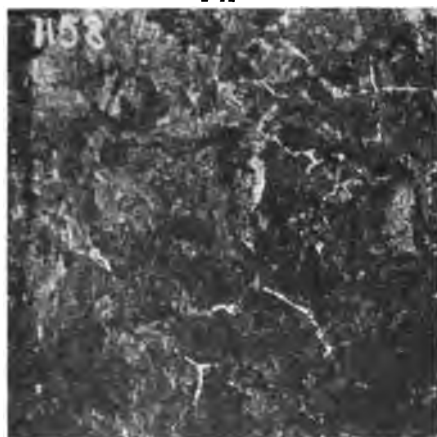
FIG. 11.—SOME OF THE RAILS AFTER BEING SUBJECTED TO DROP TESTS.



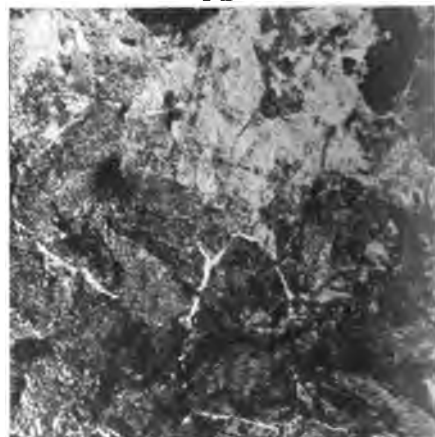
1 A



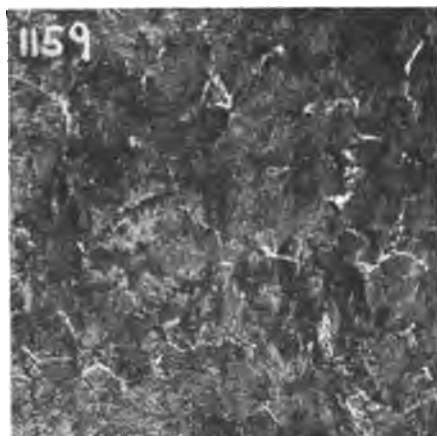
1 D



1 B



1 E

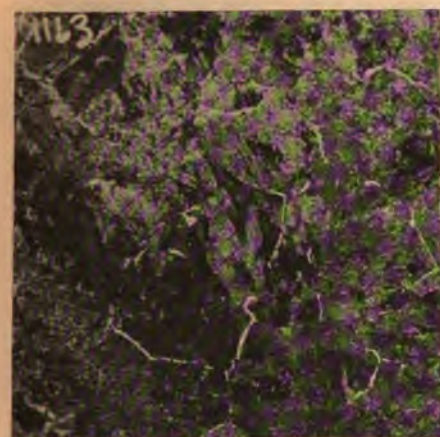


1 C

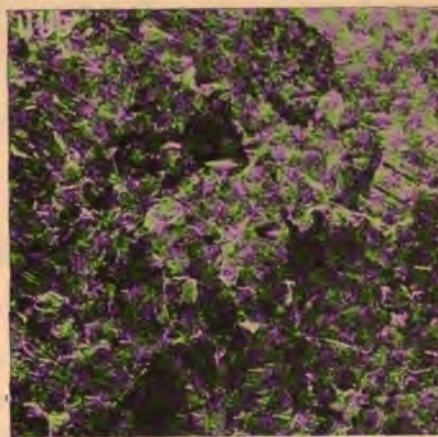


1 F

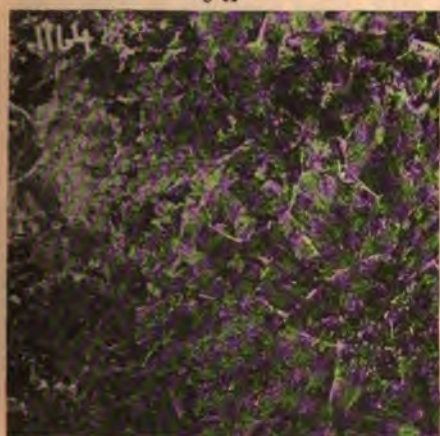
FIG. 14.—RAILS 1 A TO 1 F. ORIGINAL SAMPLES HEATED TO 1,800° F. AND FURNACE COOLED.



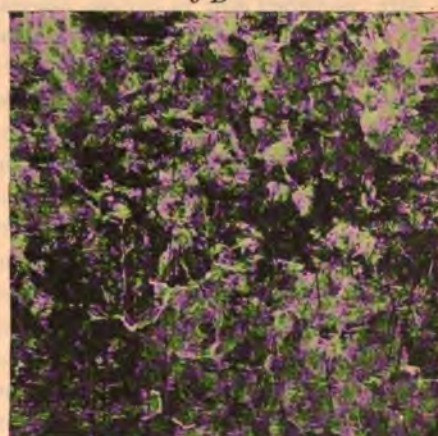
9 A



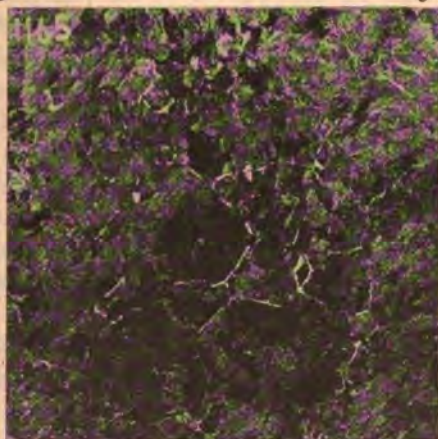
9 D



9 B



9 E



9 C

FIG. 15.—RAILS 9 A TO 9 E. ORIGINAL SAMPLES HEATED TO 1,800° F. AND FURNACE COOLED.

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Are the Deformation Lines in Manganese Steel Twins or Slip Bands?

BY HENRY M. HOWE AND ARTHUR G. LEVY, NEW YORK, N. Y.

(New York Meeting, February, 1915)

§1. INTRODUCTION.—Any given piece of metal is made up of a very great number of *grains*, usually microscopic, each of which is a perfect crystal save only in outward form, with *cleavage planes* of low cohesion, geometrically arranged quite as in the familiar non-metallic crystals.

The plastic deformation of such a mass occurs chiefly through the *slipping* of the crystalline blocks of which each grain is composed along these cleavage planes, causing steps called *slip bands* to form on a previously polished surface. In this slip, the metal immediately adjoining each of the cleavage planes along which the slip takes place is thought to pass extremely rapidly through a very *mobile* state into the *amorphous* state, in which it is harder than the remaining still crystalline metal between the amorphous layers thus formed along the slip planes. Whether the slip planes below the surface in ferrite can be detected by cutting, polishing, and etching sections is in dispute; but if this detection is possible at all, it is usually made impossible by heating the metal even gently. In short, the change along the slip planes does not persist through heating, and it is thought not to persist through time even at the room temperature, nearly all the amorphous metal re-crystallizing.

The network in Fig. 22 and the parallel N.-70°-E. lines in Fig. 21 are slip bands in copper.

When plastically deformed metal is heated, certain parts of certain grains may *twin*, that is their component crystalline units may rotate into a position, or more exactly into an orientation, symmetrical with the initial orientation and hence with that of the remainder of that grain. As seen in a microsection of a metal the twinned areas usually have parallel sides. The existence of twinning is recognized in metallic sections either by the presence of such parallel-sided areas differing in brightness from the adjoining metal, especially when seen under oblique light after etching, or more surely by the zigzagging of the slip bands which form when twinned metal is itself deformed again. Hence the usual procedure in developing twins is to deform plastically so as to cause the twinning tendency; to heat so as to allow the tendency to assert

itself; to polish so that slip bands may next be made visible, and to deform again so as to create slip bands. On this polished surface such slip bands change direction sharply on entering each twinned area, and revert to their initial direction on leaving it.

Twins made manifest in this the usual way may be called *annealing* twins, to distinguish them on one hand from congenital twins formed in solidification or transformation, which do not concern us in this paper, and on the other hand from *mechanical* twins, such as the Neumann lines are supposed to be. These are narrow bands which form in alpha iron or ferrite, usually only on deformation by shock, though we have succeeded in developing them in silicon steel by quiescent deformation in a vise. They are caused by the deformation without the need of subsequent annealing.

The parallel-sided band, 4, in Fig. 2, and the areas 1 to 7 in Fig. 1, are twins. Note how the manganese steel lines in Fig. 1 have two alternative directions, following one of them in the odd-numbered twinned areas, and the other in the even-numbered areas. And note how the slip bands in Fig. 21 change from N.-70°-E. to nearly horizontal and back again as they pass through the twinned area *abcd*.

Because twinning usually occurs in other metals, not during the plastic deformation but during a heating which follows it, and for other reasons, it cannot be regarded as the usual mechanism by means of which plastic deformation occurs, but rather as a result of deformation.

On a previously polished surface of Hadfield's austenitic manganese steel quenched from 1,000° to 1,100° plastic deformation develops lines which though they look much like slip bands, as in Figs. 1, 2, 6, 7, 8, and 10, yet differ from most slip bands not only in reappearing persistently on repolishing and re-etching as in Figs. 3, 4, 11, 12, and 15 but in persisting through considerable reheating, and indeed then becoming very much more prominent, through the precipitation of cementite along them as in Figs. 5, 9, 13, and 14.

What are these lines? According to Osmond and Cartaud¹ they are extremely numerous twins. The present paper, after outlining in Section 3 and 4 the life history of these lines, offers in Sections 13 to 17 evidence and reasoning tending to show that they are slip bands comparable with those which form in other metals and alloys. Meanwhile let us call them simply the *manganese steel lines*, so as to avoid committing ourselves to any theory by means of a more indicative name.

§2. TWINS of the usual kind have not, so far as we know, been shown hitherto in manganese steel, nor have we succeeded in developing them

¹ *Journal of the Iron and Steel Institute*, vol. lxxi, (1906, No. III), p. 468. Such multiple or "polysynthetic" twinning does indeed occur in non-metallic native minerals. Compare Iddings, *Rosenbusch's Rock-Making Minerals*, Plate 25, Fig. 6 (New York, 1889).

except by very severe deformation, followed as usual by annealing, polishing, and re-deforming. But by this means we have developed a great number of very typical twins. Two twinned fields in this alloy are shown in Figs. 1 and 2. These represent metal from the immediate neighborhood of the fracture of a tensile test piece, which after rupture was heated to $1,000^{\circ}$ for 1 hr. to develop the twins, quenched, polished, and then again deformed so as to develop the zigzagging lines. In Fig. 1 the lines follow a common direction throughout the four odd-numbered areas, and another common direction throughout the three even-numbered ones. In Fig. 2 they follow a common direction throughout the three odd-numbered areas, but not in the two even-numbered ones. The sides of these areas are in general parallel, especially those of area 4 of Fig. 2. In Fig. 1 the brightness of the nose of the tread or boundary between areas 4 and 5 and the darkness of the boundary between 5 and 6 are typical.

In strong contrast with the difficulty of causing typical twins in Hadfield's manganese steel is the great ease with which they form in an iron alloy containing 33 per cent. of manganese, and represented to be free from carbon. This was made for one of us by the Goldschmidt Thermit Co., of New York. In the etched section of a cast bar of this alloy, prepared without any deformation of which we are aware save that of sawing it open, we found large perfectly typical twins after reheating to $1,100^{\circ}$ and quenching.

§ 3. THE MANGANESE STEEL LINES.—*The progress of their development*, as traced on the previously polished surface of a wedge gently compressed,² is shown in the series of Figures from No. 16 to No. 19, Plate 4.

At the beginning of the deformation, shown in Fig. 19, there are a few parallel lines, a gentle undulating disturbance of the whole surface, and

² This method we have borrowed and modified from Frémont, *Mesure de la limite élastique des métaux*, *Bulletin de la Société d'Encouragement pour l'Industrie Nationale*, vol. cv (1903, II), p. 363. It was followed by Osmond, Frémont, and Cartaud in their classical investigation, *Les modes de déformation*, *Revue de Métallurgie*, vol. i, p. 27 (1904). These investigators used a truncated pyramid, but for simplicity we have replaced this with the truncated wedge shown in Fig. 24, $\frac{1}{2}$ in. high from base to truncated top, $\frac{3}{8}$ in. thick, $\frac{1}{2}$ in. by $\frac{3}{8}$ in. at the base, and $\frac{1}{4}$ in. by $\frac{3}{8}$ in. at the top. After polishing one of the trapezoidal faces of the wedge so that the deformation lines might be seen there, we compressed it between polished and greased steel plates, with pressure applied at the wide end or base and at the truncated top or narrow end. Strangely enough there was in most cases, in manganese steel, copper, and other metals, a band almost free from deformation, running along the upper part of the wedge, probably because of friction with the compressing plate, in spite of the greased contact between them intended specially to prevent this result. Below the region where the dulling of the polished surface ceased, and near the base, came a new region of slight deformation, the arc of a circle of long radius.

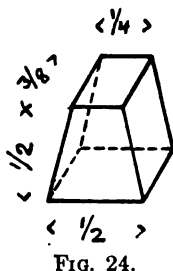


FIG. 24.

the outlining of some of the grain boundaries, all three phenomena reproducing closely what occurs in other metals except alpha iron. Then these parallel lines become more marked, and are crossed by a conjugate set, as in Fig. 18. Next they become both more numerous and more prominent, as in Fig. 17. Under still greater deformation the surface distortion or ruffling becomes so great that it tends to mask these lines, as in Fig. 16.

The parallel lines in manganese steel, and the slip bands in copper, become visible at about the region where the undulation of the surface can first be recognized, and hence under about the same degree of deformation. Judged by means of the accompanying degree of surface undulation, the manganese steel lines appear if anything a little earlier than the slip bands in copper. Even where the surface deformation is very marked there are undulated areas in which no lines can be traced under a magnification of 50 diameters, and but a few under a magnification of 500 diameters. Nevertheless the whole of the undulation may be accompanied and accounted for by slip bands so minute as to escape detection.

§4. THEIR CHARACTERISTICS.—Whether caused by shock or by quiescent deformation they are in parallel sets, sometimes in one direction only in each grain, as in Figs. 3, 7, 11, and 19, but more often in two and sometimes even in three directions, as in Figs. 4 and 5. These directions were identified by Osmond and Cartaud as representing the octahedral cleavages.³ They often recall martensite, when, as in Figs. 4 and 5, two sets of them are of about equal prominence. In their incipency they may run but part way across a grain, as in Figs. 6 and 19, but under greater deformation they usually extend completely across the grain in which they occur, as in Figs. 3, 5, and 10. Though on reaching a grain boundary they generally stop, as in Figs. 3, 5, and 10, they may continue across grain boundaries, sometimes without changing direction, as in Fig. 13, sometimes deflecting sharply, as at the boundary *ab*, in Fig. 11, and at the boundaries between grains *c* and *e*, and *f* and *g* in the same figure. They may cross the dendrites without the least deflection, as in Fig. 8. They are usually very nearly straight, or curved smoothly and gently, but occasionally they wave like lamellar pearlite, probably in cases of great deformation, as just above *e* in Fig. 11.

They very rarely fault each other sharply as slip bands in other metals and Neumann lines in alpha iron often do, perhaps because, for given deformation, they are so numerous that the fault at any one intersection is so slight as to escape detection. Indeed some at least of what seem at first to be faults on closer examination prove to be twinned areas.

A very slight deformation suffices to cause them. Thus, though they are lacking at least usually in castings as taken from the sand, they may

³ *Loc. cit.*

be made extremely prominent by the slight unintentional deformation which occurs when a casting cut almost completely through, is then broken in two.

As they form the metal becomes somewhat ferro-magnetic, and increases greatly in hardness. Thus the Brinell hardness near the tensile fracture shown in Fig. 12 was 340, whereas in the undeformed part of the same test piece it was only 217.⁴ This magnetization indicates that some of the metal is transferred to the alpha state. The hardness is to be referred at least in part to the amorphousness of this alpha iron caused by the slip and perhaps in part to the cementite which may be precipitated simultaneously as explained beyond.

Thus Fig. 12, showing the longitudinal section through the fracture of a tensile test piece, suggests that its white component is alpha and in part amorphous iron formed along the slip planes on all sides of the dark rhomboids of the residual crystalline and presumably gamma iron. A larger magnification, as in Fig. 15, tends to confirm this interpretation.

§5. THE CHIEF REASONS FOR THE USEFULNESS OF THIS ALLOY are probably its ability to retain not only continuity but strength through an extraordinarily great degree of amorphization, and the great ease with which this amorphization occurs, two aspects of the same thing. Its surface flows very easily under strong pressure, as for instance that which it meets in the jaws of stone crushers, but in this very flow it hardens greatly and rapidly, so that a relatively hard wearing surface forms, integrally united with a ductile back. Moreover, this surface is self-renewing, for as fast as a given layer is worn away the next, as it thus becomes the surface, simultaneously hardens under the deformation which it at once undergoes.

§6. THE PRECIPITATION OF CEMENTITE along these lines is evidently one cause of their apparent persistence.

The transfer of the iron from the gamma to the alpha state caused by the deformation may perhaps itself precipitate cementite, which is nearly or quite insoluble in alpha iron though abundantly soluble in gamma iron, but we cannot be certain yet that the cementite is able at so low a temperature to escape from solution. However this may be, the heating certainly should enable this cementite to precipitate, and as the temperature rises and the mobility increases, to coalesce into visible masses. This same mobility should allow both the pro-eutectoid cementite, the precipitation of which has been suppressed in the quenching, to precipitate, and the gamma iron itself to change into alpha, with consequent precipitation of the remaining cementite.

⁴ Hadfield and Hopkinson found the Brinell hardness of a tensile test piece after rupture 540, against about 220 in like material before straining. (*Journal of the Iron and Steel Institute*, vol. lxxxix (1914, No. I), pp. 112 and 124, and *Bulletin* No. 87, Mar., 1914, p. 523).

Each block of crystalline metal, which we may picture to be represented by the dark rectangles of Fig. 12, being surrounded on all sides by amorphous metal, would now act as an independent crystal, and expel to its outside all the cementite precipitated within it. This cementite should thus become assembled along the very cleavages in which the slip has created amorphous iron, and because these cleavages are octahedral, it would form the familiar Widmanstätten figuring, so prominent in Figs. 5, 13, and 14. The thickening of the manganese steel lines on reheating strained specimens is evidently due to this assembling of cementite along them, as is shown most strikingly in Fig. 14.

This crowded migration of cementite would be likely to sweep along with it any inert matter, or sonims, such as the slag and manganese sulphide so often present in this steel.

§7. CEMENTITE, NOT MARTENSITE.—This cementite was very naturally taken by Sauveur⁵ for martensite, which it often simulates closely, both of them lying Widmanstättenwise along the octahedral cleavages like the amorphous iron. It is shown to be cementite by its blackening with sodium picrate;⁶ by its assembling, under favorable conditions, into a network closely like that of the pro-eutectoid cementite of hypo-eutectoid carbon steel; and by its being apparently identical with the corresponding cementite network which forms in this manganese steel, as shown in Figs. 5 and 13.

§8. ADDITIONAL CAUSE OF THE PERSISTENCE OF THESE LINES.—If the reappearance of these lines after reheating were due solely to the assembling of cementite along them, then they ought to thin out progressively on heating to temperatures above A1 and then quenching, and to be effaced progressively on heating to above SE and quenching, so that the proportion of these lines which reappear ought to be the smaller the higher and the longer the heating is. As regards temperatures between A1 and SE our scanty observations do not disagree with this expectation. But the reappearance of these lines after heating to 1,000° and quenching, which certainly was unforeseen, indicates that their persistence is connected with something in addition to the presence of cementite, which even in this sluggish material ought to re-dissolve completely in a stay of 1 hr. at 1,000°, and to remain dissolved during the quenching if, as is probable, the line SE is below 1,000° for this material.

The persistency of this specific Widmanstätten figuring tallies with its persistency under other conditions,⁷ and with that of the initial

⁵ *Bulletin* No. 93, Sept., 1914, p. 2439.

⁶ Howe, *Trans. Faraday Soc.*, vol. x (1914); *Engineering*, vol. xcix (1915), p. 87.

⁷ We showed that this structure has extraordinary stability in a 0.40 carbon steel, suffering no important breakdown on staying either for 137 hr. at 650°, or for 21 ½ hr. at between 725° and 740°, though in this latter case the very strong cementite network structure of a 1.14 carbon steel beside the Widmanstätten steel broke down com-

dendritic casting or ingot structure,⁸ both of which persist through treatments which at first would be expected to efface them. In some cases this persistency represents simply a defiance of surface tension, in others it may be referred to the co-concentration of minute quantities of impurities such as phosphorus, which diffuses very slowly, or of sonims which do not diffuse. In this present case, the persistence of the manganese steel lines through a long sojourn at 1,000° and quenching may be due to the latter cause, lines of impurities once swept along by the migrating cementite into the cleavage planes where the amorphous iron formed, remaining undiffused long after the cementite which dragged them there has diffused away and been reabsorbed, and the amorphous state has reverted to the crystalline state, and the gamma iron to alpha.

§9. THE PERSISTENCY OF THE MANGANESE STEEL LINES.—Some of these manganese steel lines reappear after repolishing and re-etching, showing that the metal, along the planes of slip throughout the mass, has undergone some change which causes it to react with the etching reagent differently from the undisturbed remainder of the metal. Rosenhain⁹ indeed finds that in other metals, if the slip is great, it can be detected in this way, by repolishing and re-etching; but that the change thus detectable by etching effaces itself very rapidly on slight heating, and slowly even at the room temperature, whence he infers that the metal reverts spontaneously from the amorphous state caused by the deformation to the crystalline state, and thus becomes undistinguishable from the remainder of the mass. Strauss,¹⁰ too, finds that steel which has been nitrogenized by heating in ammonia, when deformed even slightly and unintentionally and then polished and etched, shows martensitic markings, which therefore should represent a detectable change induced along the slipping planes by the deformation.

But the manganese steel lines persist even through reheating. Or more accurately, after the deformed metal has been reheated, to repolish and re-etch it develops lines, shown in Figs. 5, 13, and 14, following the directions of those found immediately after deformation, as for instance those of Figs. 4, 10, 11, 12, and 16 to 19 inclusive. Indeed the lines found after reheating may be much more prominent and thicker than those found before. Note how conspicuous the white masses in Fig. 14 are after reheating, repolishing, and re-etching. On a smaller magnification, as in Figs. 5 and 11, they are seen to correspond to those

pletely. Notes on Divorcing Annealing and Other Features of Structural Coalescence in Iron and Steel, *Proceedings of the Cleveland Institution of Engineers*, July, 1914, pp. 234 to 243, and remarks by Stead on p. 252.

⁸ Engineer N. T. Belalew, *Révue de Metallurgie, Mémoires*, vol. ix, No. 9, p. 647 (Sept., 1912).

⁹ Metals, Crystalline and Amorphous, *Engineering*, vol. xevi, No. 2493, p. 510 (Oct. 10, 1913).

¹⁰ *Stahl und Eisen*, vol. xxxiv, No. 50, p. 1814 (Dec. 10, 1914).

found after etching the deformed metal without heating it, as in Fig. 4; and these in turn are seen to correspond to those which form on the polished surface during the deformation itself, as in Fig. 7.

The temperatures after heating to which these lines reappeared on repolishing and re-etching in our experiments are 400°, 550°, 575°, 600°, and 800°. Some of these lines could be traced even after a stay of 1 hr. at 1,000° followed by quenching, polishing, and etching. It was not the quenching that caused them in this case, because we find that polishing and etching does not develop them in a specimen quenched from 1,000° without previous deformation.

That they persist at the room temperature is to be referred to the formation of alpha iron along them, which of course has no tendency to revert to gamma iron. But, on heating, this gamma iron itself ought to change over into alpha, and the amorphous iron ought to revert to the crystalline form, so that here another cause must be sought for the persistency of these lines.

§10. CEMENTITE AND RUPTURE.—This pro-eutectoid cementite network forms the path of rupture in manganese steel, as is shown in Fig. 9, even more prominently than in hyper-eutectoid carbon steel, and is probably extremely embrittling in both alloys.

§11. GROUPINGS WHICH SIMULATE TWINNING.—There are two distinct sets of groupings of these manganese steel lines which at first look like twinning, but on further examination may prove not to be. The first set represents the intersection of these lines at grain boundaries, the second represents an alternation of the relative prominence of two conjugate sets of these lines.

§11A. INTERSECTIONS AT GRAIN BOUNDARIES.—In Fig. 11 the intersection *ab* of two sets of lines certainly looks like a boundary between two twinned areas, and not between two adjoining grains, for not only is the proportion of the lines which continue past *ab* from one area into the other much greater than the proportion of slip bands which usually cross a grain boundary, but also the change of direction is much greater than is usual in slip bands which cross grain boundaries. Yet the shape of the two areas on either side of *ab*, while very far from that of the parallel-sided twin areas, is exactly that of the usual grain areas. This is even more strikingly true of such areas as *c* and *d*, and of *f* and *g*, yet these manganese steel lines cross the boundaries between each of these pairs of areas with a marked deflection. Because so many of these areas are thus exactly grain shaped, and because this alloy has occasional twinned areas with the characteristic parallel-sided twin shape, and wholly unlike these grain-shaped areas, we should take these latter for grains, and infer that in this alloy an unusually large proportion of the slip bands continue across the grain junctions even when, as at *ab*, this implies an apparent great change of direction, remembering that the apparent change depends

wholly on the inclination of the plane of the section to the planes of slip.¹¹

§12. TWINNING SIMULATED BY A CHANGE IN THE RELATIVE PROMINENCE OF TWO CONJUGATE SETS OF MANGANESE STEEL LINES.—Both in the polished but unetched and in the polished and etched sections of plastically deformed manganese steel, there are very often two conjugate sets of lines, certain individuals of which stop or lose prominence on intersecting a line of the conjugate set, or lose prominence on entering a certain area. Thus in Fig. 6 two lines running from *c* to *d* at first look as if they stopped at *d* and changed direction, whereas in fact they continue their direction down beyond *d*, but become fainter at the left of *d*. So the lines running S.-60°-W. from *f* at first seem to stop at *e* and change direction, whereas in fact they extend beyond *e* but lose

¹¹ Humfrey indeed asserts that slip bands in ferrite do not cross grain boundaries unless they can do this without change of direction (*Carnegie Scholarship Memoirs, Iron and Steel Institute*, vol. v, 1913, pp. 91, 92). But his Fig. 7, Plate XI, shows at least four slip bands which deflect more than 90° in crossing a grain boundary. Moreover, the apparent angle of deflection must need vary greatly with the direction of the plane of the section, as is shown by a very simple experiment, which we commend to our readers as helping them to grasp these general conditions. Cut out of a sheet of stiff paper an isosceles-triangle arrow head, the two sides of which meet at an angle of about 70° like the lines on either side of the intersection *ab* of Fig. 11. Set an open book on end on your desk, insert the point of the arrow head, with its base horizontal, between the leaves, and open or shut the book till the arrow head just fits the opening between its leaves. If the bisectrix of the triangle is held horizontal, the leaves will of course fit the sides of the arrow head when they meet at this same angle, 70°. If the book is now opened gradually wider and wider, we can continue to make the sides of the arrow fit the space between the leaves by simply raising its point and lowering its base, theoretically till the book is open wide so that its two covers are in the same plane.

Now let the arrow head represent the plane of our microsection, and the leaves of book represent the direction of two slip planes which approach each other across adjoining grains, and intersect at the grain boundary, the back of the book. Let the book be nearly wide open, the angle between its covers being 170°. The sides of an arrow which if held horizontally is to fit between these leaves would have to meet at this same obtuse angle, 170°. If, now, the arrow is inclined more and more by raising its point while keeping its base horizontal, it will have to be trimmed off first so as to be less obtuse, then so as to be a right angle, and then to a more and more acute angle, in order to remain always just fitting in between the leaves, which remain at an angle of 170°. If we replace the arrow by the plane of our microsection then if we incline it more and more from the horizontal, the angle at which the leaves will cut it will in like manner become first less obtuse, then a right angle, and then a more and more acute angle. In short, the apparent angle between the slip planes on the two sides of a grain boundary varies greatly with the inclination of the plane of the microsection to those planes. Thus in two adjoining grains A and B, two slip planes, which in fact are within 10° of being parallel so that slip along one of them in grain A might easily cause slip along the other in grain B, if cut by a microsection which is strongly inclined to both may seem to make a very sharp angle, so that the continuation of slip bands across the grain boundary looks as if the direction of slip changed by say 100° as at *ab* in Fig. 11 though in fact it changes by only 10°.

prominence. Again, at the imaginary line *ab* a great number of the lines running S.-60°-W. seem to twin into corresponding lines running N.-45°-W., as if *ab* were the boundary of a twinned area. Yet if we extend *ab* a little higher up we see that these two sets of lines in fact continue beyond their intersection. So in the etched Fig. 4 certain areas running N.-80°-E. seem to be twinned areas, for in the first, third, fifth, etc., the prominent lines run N.-40°-W. whereas in the second, fourth, sixth, etc., they run N.-40°-E. But on closer examination these two sets of lines, N.-40°-W. and N.-40°-E., are seen to co-exist in a larger part of each of these areas, with the difference that in the odd-numbered areas the N.-40°-W. lines are the more prominent, whereas in the even-numbered areas the N.-40°-E. lines are. In short, though these cases certainly suggest strongly the true staircase twinning effect of Fig. 1, in which lines running in a given direction change abruptly to another direction on entering a parallel-sided area, yet the co-existence of both sets of these lines in a large part of these areas argues that these are cases not of twinning substitution of one direction for another, but only of a change in the relative prominence of two sets of lines.

§13. ARE THE MANGANESE STEEL LINES TWINS OR SLIP BANDS?—We find five reasons for interpreting them as slip bands.

(1) With progressively increasing deformation, the first appearance and the development of these lines are like those of the slip bands in other metals, as we find by applying to iron, copper, and lead the truncated wedge experiment outlined in Section 3. This resemblance is shown by a comparison of Figs. 16 to 19, which illustrate the development of these lines in manganese steel, with Figs. 20 to 23, which illustrate the development of the slip bands in copper.

Save that copper roughens more than manganese steel, one metal might often be taken for the other, so closely do the manganese steel lines and the copper slip bands correspond in their development. Compare for instance Figs. 18 and 22. The manganese steel lines curve sharply at their intersections like those of copper, and they and the copper lines deviate from straightness in about the same degree, as is seen by holding a straightedge along lines *ab* of Fig. 18 and *a'b'* of Fig. 22.

These lines in copper are certainly slip bands. Those in manganese steel start and develop as closely like those in copper as could be in two different metals. The differences are rather such as we expect between identical phenomena in different substances, such as the difference between the waves in oil and those in water and sulphuric ether, than such as we find between radically different phenomena.

§14. (2) These lines in manganese steel bear to the twinned areas the same relation as regards their order of magnitude that slip bands bear to twinned areas not only in copper, silver, and lead, but also in 25 per cent. nickel steel, the substance most closely comparable with manganese

Manganese Steel

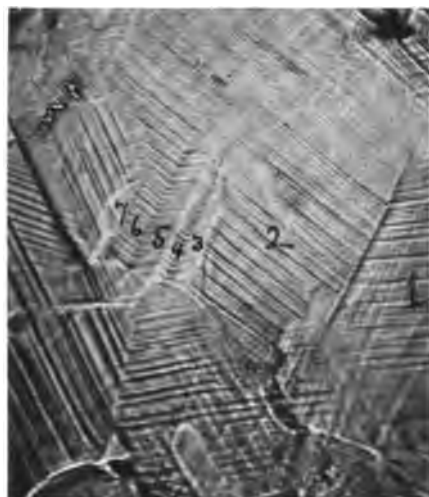


FIG. 1. × 500

Twins in Mn steel, greatly deformed, annealed, polished, and hammered. Not etched.

FIG. 2. × 500

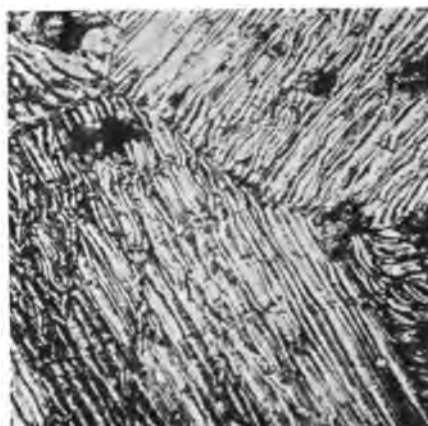


FIG. 3. × 500

Mn steel lines in one and the same casting, very deeply etched.

FIG. 4. × 500

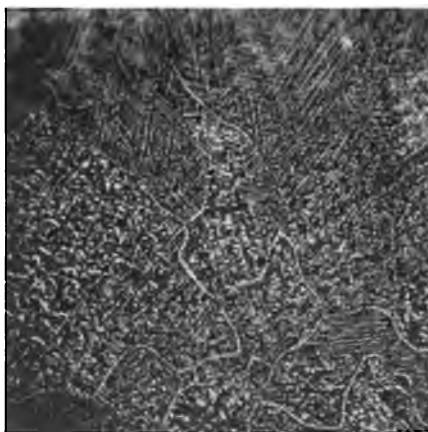
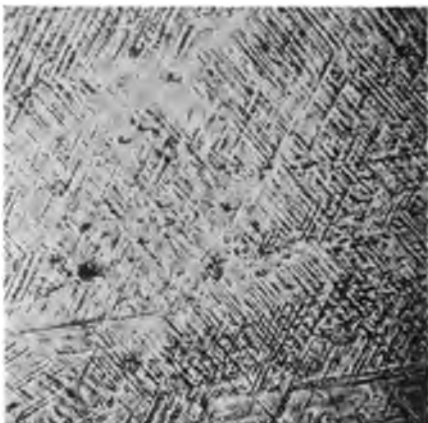


FIG. 5.—Cementite in Mn steel, strained, heated to 575° C., polished, and lightly etched. × 50.

Manganese Steel



FIG. 6.—Lines in lightly strained Mn steel, simulating twins. Not etched. $\times 500$.



FIG. 8.—Lines in Mn steel severely strained, crossing the dendrites. Not etched. $\times 50$.

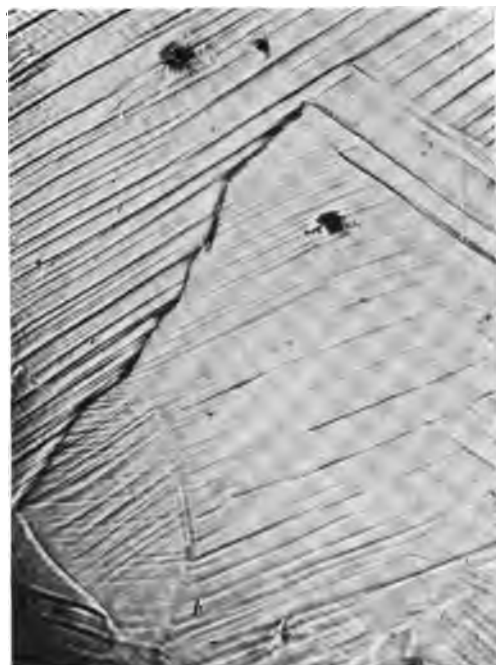


FIG. 7.—Lines in Mn steel stop at grain boundaries. Not etched. $\times 500$.



FIG. 9.—Rupture follows the cementite in Mn steel reheated to 575°C . Etched lightly. $\times 500$.

Manganese Steel

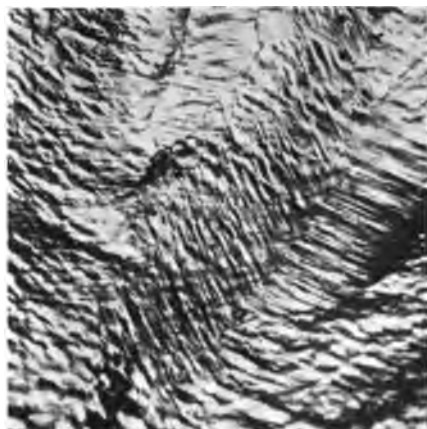


FIG. 10.—Columnar crystals disclosed by hammering an unetched polished surface. $\times 50$.

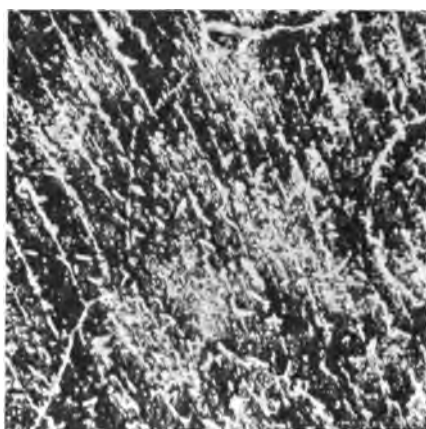


FIG. 13.—Mn steel lines cross grain boundaries. Same treatment as Fig. 14. Etched lightly. $\times 180$.

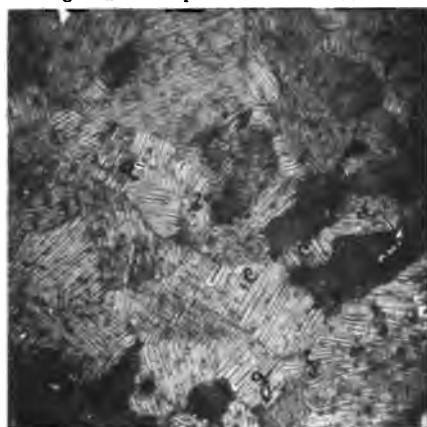


FIG. 11.—Small casting strained and etched deeply. $\times 50$.

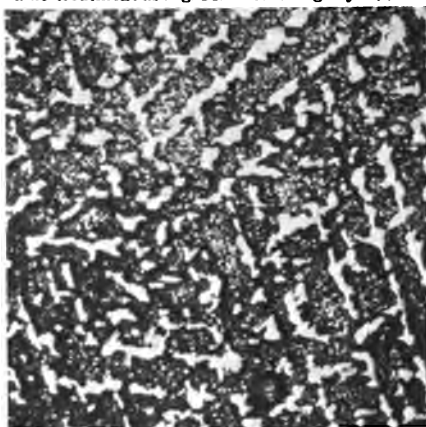


FIG. 14.—Cementite along slip planes. Strained, heated to 575°C ., etched lightly. $\times 500$.

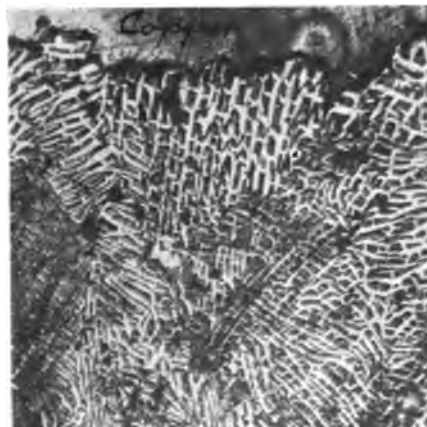


FIG. 12. $\times 90$.

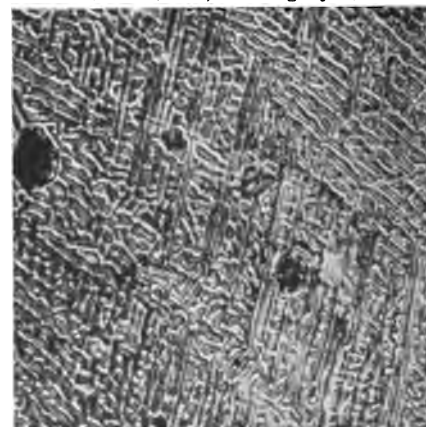
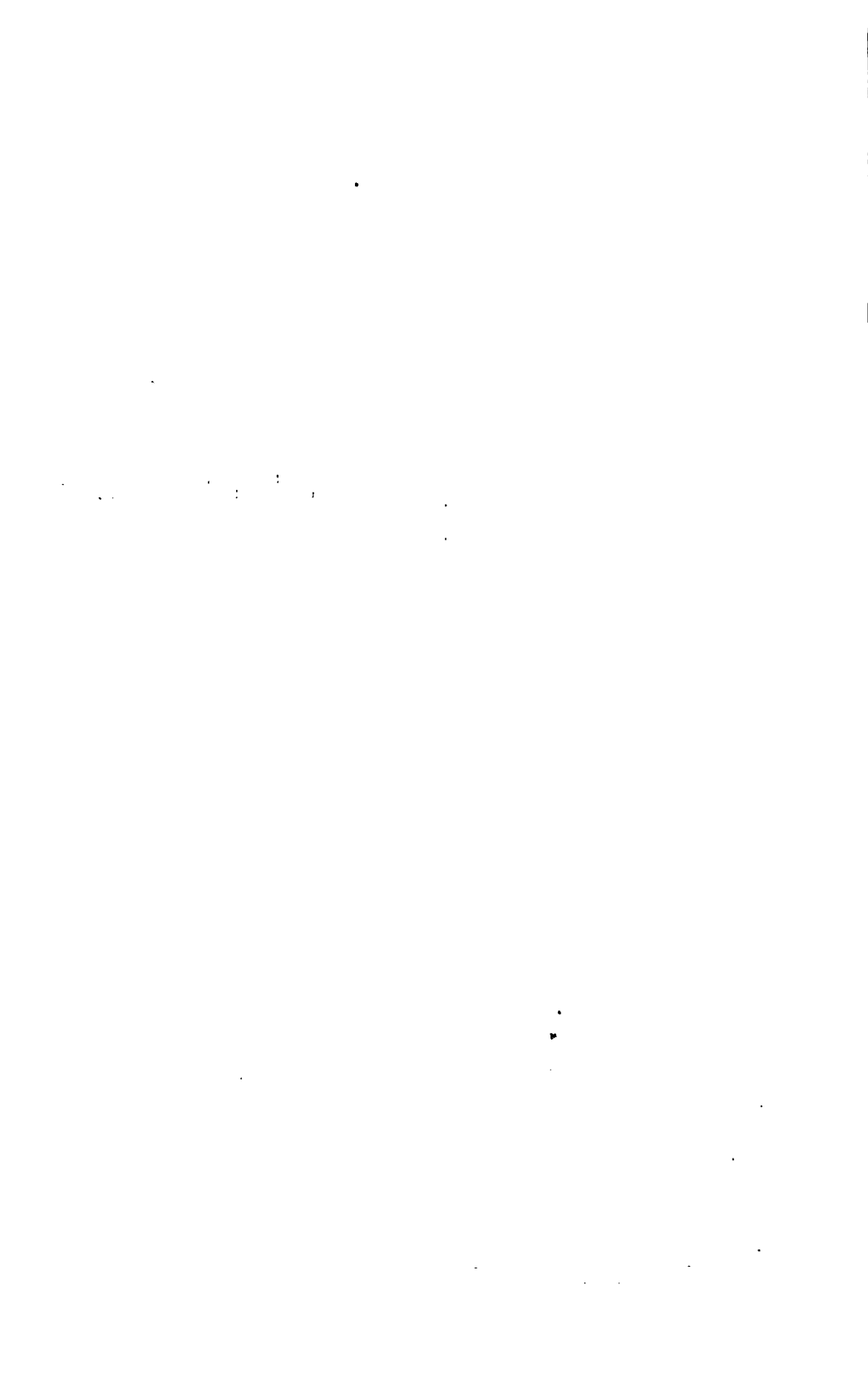


FIG. 15. $\times 500$

Longitudinal section through copper-plated tensile fracture. Etched deeply.



Progress of the development of the lines in Mn steel and of the slip bands in copper.

Manganese Steel

Copper

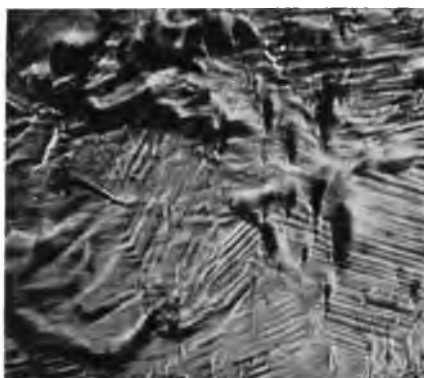


FIG. 16.

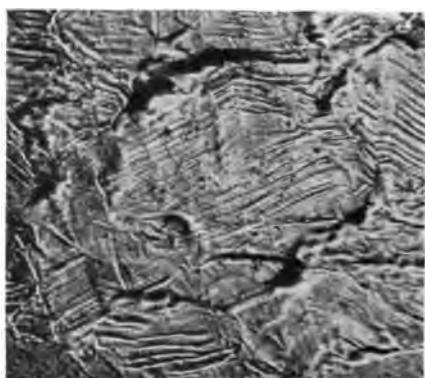


FIG. 20.

Great deformation. $\times 180$.

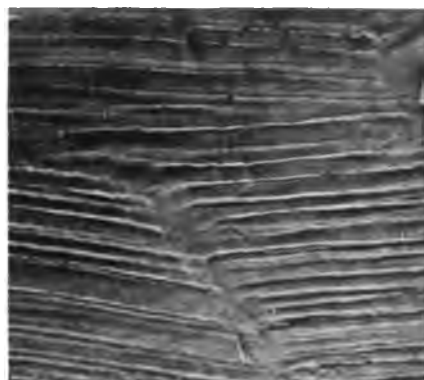


FIG. 17.



FIG. 21.

Less deformation. $\times 500$.

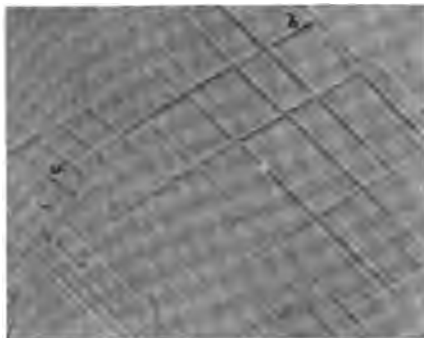


FIG. 18.



FIG. 22.

Still less deformation. $\times 500$.

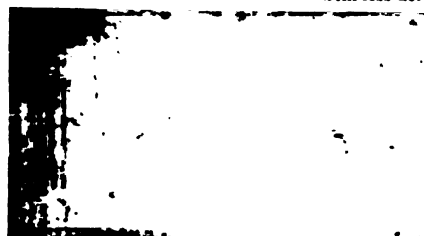


FIG. 19.



FIG. 23.

Least deformation. $\times 180$.

steel, differing from it primarily in being niccoliferous instead of manganiferous austenite, and in not containing dissolved cementite. Indeed, these lines and the twinned areas in Fig. 1 might easily be taken for slip bands and twinned areas in copper.

§15. (3) The conditions under which the lines form in manganese steel are the same as those under which the slip bands habitually form in these other metals, and differ radically from those under which twinning usually occurs. These lines like the slip bands result directly from deformation, and thus are mechanical, whereas twinning is rarely mechanical but usually can be induced only by annealing after deformation, except in the extremely soft self-annealing mobile metals such as tin and lead. And though the alleged twins of alpha iron, the Neumann lines, can be caused mechanically and without subsequent annealing, their creation usually requires shock, whereas quiescent stress creates the manganese steel lines as it does the slip bands of all other metals.

§16. (4) So far as we know, slip bands can be developed readily without twinning, but the stress which causes twinning must also cause slip bands, which represent the normal and easy mode of translation. Thus the quiescent deformation which creates the slip bands so abundantly in alpha iron gives no suggestion of Neumann lines, whereas the shock which develops the Neumann lines always develops slip bands also and nearly always abundantly. Even in coarsely crystalline steel in which the Neumann lines form very readily, they are always accompanied by some slip bands. Hence when, as in manganese steel, only a single set of lines forms, analogy indicates that these lines are more likely to be slip bands than twins. This inference is not materially weakened by the fact that certain areas containing Neumann lines may be nearly or quite free from slip bands, and *vice versa*.

§17. (5) The amorphization of the metal along the internal planes of which these manganese steel lines are the outcrops, testified to by the hardening, tallies with the belief that slip has occurred along them but not with the belief that only twinning has. For slip is naturally believed to cause such amorphization, whereas twinning is not, because it implies only rotation into a new orientation without crushing any crystalline matter.

§18. Another consideration, which at first seems to argue that these lines must be slip bands, on examination proves fallacious. That consideration is that in other metals we refer the deformation primarily to slip bands, first because the twinning does not occur during the deformation, and second because for other reasons it seems incompetent to explain the deformation. This might suggest at first that, if the manganese steel lines are all of one kind, they must be slip bands in order to explain the deformation. But these manganese steel lines are competent to explain the deformation even on the theory that they are twins, first because they

form during and not after the deformation, and second because they are numerous enough to account for great deformation. For, if we divide them mentally into the odd and the even numbered, and assume that the odd, for instance, are thicker than the even, then the difference would be cumulative from pair to pair.

§19. THE PERSISTENCY OF THESE LINES DOES NOT INDICATE THAT THEY ARE TWINS.—Because in other metals the slip bands have but slight persistency, as already pointed out, whereas twins are much more persistent, we might at first infer from the great persistency of the manganese steel lines that they too are twins. But the radical difference in conditions removes all reason for such an inference. The transitoriness of the slip bands in other metals means that the change which occurs along their slip planes either is not detected by any reagent yet tried, or as Rosenhain holds is effaced quickly on reheating and slowly even in the cold because the amorphous state is here the unstable and the crystalline state the stable one.

But in manganese steel the change from gamma to alpha along the slip planes, proved by the magnetization which accompanies deformation, cannot reverse itself in the cold, because alpha is here the stable form.

The reappearance of these lines or modification of them after gentle reheating we have already referred to the precipitation of cementite along them, and after high heating to the persistence of undiffusing impurities assembled along with the cementite.

§20. OSMOND AND CARTAUD ON THESE LINES.—Against these reasons for regarding these lines as slip bands we have the belief of Osmond and Cartaud¹² that they are twins, apparently because they are persistent, though we cannot be sure that this was the determining cause of that belief. There is the further suggestion that they found these lines related rather to the twins than to the slip bands which form in chrome-nickel steel. We believe that their assertion may be set aside provisionally, not so much because they fail to indicate in what respect these lines resemble twins rather than slip bands, as because they were very probably unaware of the evidence and the reasons outlined in this present paper and especially of the reasons why slip bands in manganese steel should persist.

§21. *In brief*, we believe that these lines may be accepted provisionally as originating in slip rather than in twinning, and as being initially slip bands representing slip along cleavages about which cementite and solims concentrate on heating.

§22. SUMMARY.—After outlining in Section 1 certain features of plastic deformation bearing on the question at issue, we record in Section 2 that we have developed in this alloy typical twins of the usual kind.

¹² *Journal of the Iron and Steel Institute*, vol. lxxxi (1906, No. III), p. 468.

We outline and in part explain the life history of the lines in manganese steel (Sections 3 and 4). They are apparently persistent (Section 1). The persistency on heating¹³ is referable to the concentration of cementite and perhaps of sonims in the very cleavages along which the slip has occurred (Sections 6 and 8). This concentration is of cementite, not of martensite (Section 7).

Both the intersections of these lines at the grain boundaries; and the alternations in the prominence of conjugate sets of these lines, may simulate twinning (Sections 11 and 12).

Indications that these lines are not twins but slip bands are that they begin and develop like slip bands in other metals (Section 13); that they bear to the twins in manganese steel the same relation as regards order of magnitude that the slip bands bear to the twins in other metals (Section 14); that the conditions under which they form are those under which slip bands form in other metals, but unlike those under which twins form (Section 15); that in other metals deformation always causes slip bands but may or may not cause twins, and hence that the fact that these lines always form when manganese steel is plastically deformed whereas the typical twins only rarely do argues by analogy that they are slip bands (Section 16); and that the amorphization which accompanies their formation would naturally accompany the formation of slip bands but not that of twins (Section 17).¹³

It is admitted that twins would be competent to explain the deformation (Section 18).

The fact that twins are far more persistent than slip bands in other metals does not indicate that because these lines are persistent they are twins, because in this alloy their creation leads to the more stable alpha form which cannot revert to gamma, whereas the amorphous metal which forms along the slip planes in other metals and alloys is the less stable, always tending to revert to the crystalline state and thus to efface the slip bands (Section 19).

The opinion of Osmond and Cartaud that these lines are twins is set aside, less because they give no readily controlled reasons for their opinion than because they were probably unaware of the evidence and reasoning here set forth (Section 20).

ACKNOWLEDGMENT.—The investigations on which this paper is based were made in the metallurgical laboratories of Columbia University, in part under a grant from the Carnegie Institution of Washington, and on manganese steel castings kindly given by Knox Taylor, President of the Taylor-Wharton Iron & Steel Co., and by the Goldschmidt Thermit Co.

¹³ Though we use for convenience the phraseology of Beilby's amorphous theory we hold that the failure of the slip bands in other metals to reappear on repolishing opposes it. This failure cannot be referred to the recrystallization of the amorphous metal, because the other effects of the alleged amorphization, such as the hardening and lightening, persist and even increase with time and slight heating.

Description of Micrographs

Fig. No.	Plate No.	Magnification	Material	Treatment	Description
1	1	500	Mn steel	Metal close to the tensile fracture. Heated to 1,000° for 1 hr., quenched. Polished, hammered, not etched.	Multiple (polysynthetic) annealing twins in greatly deformed manganese steel with typical sagging of lines.
2	1	500	Mn steel		Parallel-sided annealing twinned area (4) in greatly deformed manganese steel.
3	1	500	Mn steel	Very small casting (link). Deeply etched.	Manganese steel lines stop short at grain boundaries.
4	1	500	Mn steel		Widmanstätten figuring of manganese steel lines in a casting only 3/64 in. thick in part.
5	1	50	Mn steel	Heated to 1,100°, quenched, strained. Reheated to 575° for 2 hr., etched lightly.	Cementite as Widmanstätten or cleavage figuring and as network in grain boundaries.
6	2	500	Mn steel	Heated to 1,100°, quenched. Polished, lightly strained, not etched.	Alternations in the relative prominence of two conjugate sets of manganese steel lines suggest twinning.
7	2	500	Mn steel	Same as Figs. 1 and 2.	Manganese steel lines stop short or change prominence at grain boundaries.
8	2	50	Mn steel	Heated to 1,100°, quenched. Polished, hammered, not etched.	Manganese steel lines cross the dendrites and probably some grain boundaries without bending.
9	2	500	Mn steel	Same as Fig. 8, reheated to 575° for 2 hr., etched lightly.	Rupture follows the intergranular proeutectoid cementite boundaries.
10	3	50	Mn steel	Casting. Polished, hammered, not etched.	The columnar grains in manganese steel made evident on a polished surface by hammering.
11	3	50	Mn steel	Same as Figs. 3 and 4.	The manganese steel lines deflect at grain boundaries so as to suggest twinning.
12	3	90	Mn steel	Extreme end of tensile fracture (copper plated), longitudinal section, etched deeply.	White alpha iron (?) network caused by great deformation, and surrounding dark kernels of gamma iron.
13	3	180	Mn steel	Heated to 1,100°, quenched. Reheated to 575° for 2 hr., quenched. Etched lightly. Hardness = 364.	The manganese steel lines thickened by cementite are nearly or quite parallel in adjoining grains and sometimes cross the grain boundaries.
14	3	500	Mn steel	Same as Fig. 9 before breaking.	Thick coalescence of cementite along the cleavage planes which the manganese steel lines followed.
15	3	500	Mn steel	Same as Fig. 12.	Detail of Fig. 12.

PROGRESS OF DEFORMATION SHOWN ON THE POLISHED SURFACE OF A TRUNCATED WEDGE OF MANGANESE STEEL

16	4	180	Mn steel	Heated to 1,100°, quenched, cut into wedge-shaped piece. Polished, squeezed in vise, not etched.	Greatest deformation, lines partly obscured by ruffling.
17	4	500	Mn steel		Less deformation, larger magnification. Many and very prominent lines.
18	4	500	Mn steel		Still less deformation. Two conjugate sets of lines, somewhat curved by faulting (?).
19	4	180	Mn steel		Incipient deformation. A few parallel lines.

PROGRESS OF DEFORMATION SHOWN ON THE POLISHED SURFACE OF A TRUNCATED WEDGE OF COPPER

20	4	130	Copper	Rolled bar of pure Lake Superior copper, cut into wedge-shaped piece. Polished, squeezed in vise, not etched.	Great deformation. The lines obscured by ruffling.
21	4	500	Copper		Less deformation, larger magnification. Many typical slip bands, with twin area <i>abcd</i> .
22	4	500	Copper		Still less deformation. Two conjugate sets of slip bands.
23	4	180	Copper		Incipient deformation. A few parallel slip bands, and fragment of a grain boundary.

The specimens shown in Figs. 5, 6, 8, 9, 13, 14, and 16 to 19 were cut from cast bars containing 12.8 per cent. of manganese, and 1.24 per cent. of carbon. The specimens shown in Figs. 1, 2, 12 and 15, cut from the tensile test piece, are thought to have the same composition.

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Metallurgical Practice in the Porcupine District*

BY NOEL CUNNINGHAM, DOUGLAS, ARIZ.

(New York Meeting, February, 1915)

MANY excellent descriptions of the mills of the Porcupine district have been written, but no discussion exclusively devoted to the metallurgical technology has been given. These notes are intended to cover this feature briefly. They are based upon 2½ years' mill operation in the district—i.e., practically since the beginning of metallurgical operations.

Character of the Porcupine Ore.—There is no oxidized ore in the district, the surface having been deeply planed by glacial action in recent geologic time. The precious-metal content is about in the proportion of 85 of gold to 15 of silver by weight; hence, the silver is practically negligible. There are two classes of Porcupine ore, having very different characteristics; these will be referred to throughout this paper as Class A and Class B.

Class A ore is a pure quartz with inclusions of schist. Generally it is heavily fractured and breaks down readily to sharp, hard grains, about minus 10 plus 20 mesh, requiring further comminution to release the gold. It carries very little pyrite; the gold is entirely free and apt to be coarse, but often spongy, going into solution readily on that account. This gold is 60 per cent. to 85 per cent. free milling, depending on the grade of ore.

Class B ore is an iron silicate schist, strongly laminated, carrying 4 to 5 per cent. pyrite; its specific gravity is 2.8 to 3.0, depending upon the amount of mineralization. In breaking the ore in the mine, generally over 25 per cent. of material through a ½-in. ring is made; the ore readily breaks down in milling and makes a comparatively large amount of non-crystalline slime; owing to its high specific gravity, however, it is quick settling. In my opinion, the gold in this ore is free, but so finely divided that it will neither pan nor amalgamate; it appears to be disseminated through the rock and not chiefly associated with the pyrite.

Veins of Class A ore occur with or without side walls of Class B, and veins of Class B occur unassociated with Class A; more often the veins are closely banded, Class A and Class B alternating, generally with Class B in excess. Both classes of ore are more or less blocky at times, and with reference to Class B this is indicative of low gold content.

* Presented also, by joint agreement, before the Canadian Mining Institute.

From a treatment standpoint neither class of ore introduces any important difficulty, although there seems to be a tendency toward reprecipitation, due probably to some element in Class B material. Practically no cyanicides are present in the ore, chemical consumption being about 0.2 lb. of cyanide per ton of ore; 1-lb. cyanide solution is sufficient for extraction, and protective alkalinity may be carried very low. With a well-designed battery and tube mill installation, a stamp duty of 15 tons or better can be readily maintained.

Outline of Treatment and Development at the Principal Mills

Although excellent descriptions of the mills and treatment methods of the Porcupine district have appeared in the technical press, it will be of benefit to outline the treatment in the five principal mills of the district in chronological order, and to comment briefly on the metallurgical trend indicated.

First Hollinger Mill.—Destroyed by fire before ready to operate. Treatment intended: Fine crushing, plate amalgamation, and concentration of tailing.

First McIntyre Mill.—Designed for treating Class A ore. Crushing by 10 light stamps, fine grinding, plate amalgamation, and concentration of tailing from amalgamation. As the mine developed, chiefly Class B ore was produced, from which an extraction could not be made by amalgamation. This mill was shut down after about a year's run.

Vipond.—Mill of 100 tons capacity treating a mixture of Class A and Class B ore. Treatment: Fine grinding and plate amalgamation. Simple amalgamation did not make a satisfactory recovery of the gold and the mill was shut down after a few months' run. Recently the mill resumed operation, amalgamation having been abandoned and a cyanide plant added. Treatment: Fine grinding in cyanide solution, agitation, and complete counter-current decantation.

Dome.—A 40-stamp mill, recently increased to 80 stamps, treating a mixture of Class A with a less amount of Class B ore. Treatment, *at start*: Stamping in water, primary amalgamation, fine grinding, secondary amalgamation, dewatering, agitation in cyanide solution, Merrill filters, to waste. *Later*: Stamping, tube milling, and plate amalgamation in water, cone classification to three products; (a) slime, dewatered and agitated in cyanide solution, Merrill filters to waste; (b) sand, leached; (c) concentrate, reground in tube mill in closed circuit with classifier and amalgamation plate, classifier overflow to slime treatment.

Hollinger.—A 40-stamp mill, recently increased to 60 stamps, treating a mixture of Class A and Class B ore, the latter predominating. Treatment, *at start*: Stamping in solution, fine grinding, concentration, concentrates amalgamated in solution and returned to table tails, table tails to

agitators, to Moore filter, to waste. *Later:* Stamping in solution, fine grinding and concentration, concentrates agitated in strong solution, washed and impounded; table tails to two steps of continuous decantation, to filters, to waste. *Now building:* Plant for agitation and complete counter-current decantation for one-third of the tailing from table concentration.

Porcupine Crown.—Plant with 10 light stamps later increased to 20, treating Class A ore entirely. Treatment, *at start:* Stamping and fine grinding in water, followed by plate amalgamation. *Later:* Stamping and fine grinding in solution, with plate amalgamation in closed circuit with tube mill and classifier, followed by agitation and complete counter-current decantation.

Second McIntyre Mill.—Plant of 150 tons capacity recently increased to 300 tons, treating a mixture of Class A, with large preponderance of Class B ore. Treatment, *at start:* Fine grinding in solution, agitation, Burt filter to waste. *Later:* The capacity was increased from 150 to 300 tons, the treatment being unchanged except that continuous decantation replaced filtration in the new unit.

Acme Mill.—Now building with 40 stamps, to treat a mixture of Class A and Class B ore, the latter preponderating. Treatment: Stamping and fine grinding in solution, agitation, concentration, concentrates to be re-ground in solution, agitated, washed and impounded, table tails to be treated by decantation.

Analysis of the Milling Practice

A tendency toward extensive alteration in treatment methods will be at once apparent from a consideration of the above outline of milling practice and development. This is chiefly due to the fact that large bodies of Class B ore are now being developed and treated, whereas the design of most of the mills was determined almost entirely from tests upon Class A ore. The entire failure of straight amalgamation is obvious. Amalgamation in conjunction with cyanidation is practiced only at the Dome, where large bodies of Class A ore are yet to be treated, and at the Porcupine Crown, where to date only Class A ore has been found. At the latter mill, however, only a small plate area is used in the classifier-tube mill closed circuit, as the ore contains a large amount of coarse free gold which is readily caught at this point.

The equipment of the Porcupine mills offers good opportunities for comparison of various machines for doing the same work.

Stamps versus Rolls and Hardinge Ball Mills.—At the Vipond and the McIntyre mills, rolls and ball mills are doing the work done by stamps at the other mills. The ore is chiefly soft schist and the ball mills have been entirely satisfactory; power per ton of ore ground appears to be

slightly higher than with stamps for the production of identical results. Steel consumption is about the same, the stamps perhaps having a shade the better of the argument in this respect; cost of operation and repairs is in favor of the ball mill, while first cost and uniformity of operation (what might be termed lack of operating "grief") are decidedly in favor of the ball mill. While my own experience in the district has been entirely with stamps, and their performance was satisfactory, I am of the opinion that the ball mill is preferable for breaking down the Porcupine ore ahead of the tube mills.

It may be of interest to note in passing that at the McIntyre a first-class Chilean mill was discarded in favor of the ball mill after the two had run side by side for a year. At the Vipond, Hardinge pebble mills are used for fine grinding, while cylindrical mills are in use in all the other mills. I do not know how the conical mills compare with the cylindrical mills in first cost, power required, performance, etc., but on theoretical lines I favor the cylindrical mill, where coarse gold is to be dissolved in solution, as more effectively trapping the gold particles and wearing them down to microscopic fineness, owing to the vertical end of the mill.

Stamping in Water and Amalgamating versus Stamping in Solution with No Amalgamation.—Probably the best opportunity for studying this point is afforded by a comparison of the Dome and Hollinger practice. The advantages claimed for stamping in water and amalgamating are a better recovery of the coarse free gold and the saving in treatment cost, due to a smaller amount of solution to precipitate and a smaller amount of precipitate to handle. At first sight it would also appear that a saving in dissolved losses from the filters would be made, owing to the smaller amount of gold in solution going to the filters.

While at the Hollinger there is a smaller percentage of amalgamable gold in the mill heads than at the Dome, on the other hand, the coarse free gold per ton of ore in the mill heads is about the same due to the fact that the head assay is nearly triple that of the Dome. Hence, if amalgamation is necessary in order to assure the dissolution of coarse gold in cyanide solution, difficulty from this source should be experienced at the Hollinger, where crushing is in solution with no amalgamation.

This is not the case, however; all the coarse gold goes into solution in the classifier-tube mill closed circuit. This is proved by two facts in connection with the Hollinger operations. Table concentration after fine grinding is practiced at the Hollinger, and any coarse free gold passing the tube mill-classifier closed circuit would be caught on the tables. No coarse gold is present in the table concentrates, however, no color of free gold ever showing on the tables; also practically no amalgam was produced (under 1 per cent. of the total values recovered) during six months, pan-amalgamation of concentrates from the tables.

Facts also indicate that crushing in solution without amalgamation

has the best of the argument in regard to amount of solution precipitated and dissolved gold mechanically lost. With the head assay at the Hollinger about three times as high, the precipitation ratio is, I understand, only about twice that at the Dome, and the mechanical loss of dissolved gold and cyanide only about one-half. Nor is any saving in dissolved losses or treatment cost proved in favor of crushing in water followed by amalgamating. On the other hand, there are added to the treatment cost, (1) cost of amalgamation, (2) cost of increased cyanide consumption due to "waste" solution precipitated and thrown away, and (3), in winter the cost of heating to mill temperature the quantity of water—equalling several times the weight of ore treated and introduced at nearly a freezing temperature—to replace the water and waste solution discharged with the tailing, which would not be necessary if crushing were done in solution. The loss of gold left in "waste" solution after precipitation must also be added to the cost.

In comparing Dome and Hollinger metallurgical practice, it is only fair to state that since the Dome ore is harder and more compact, the gold may be less spongy and therefore less amenable to cyanidation. The only deduction which can be drawn from the facts, as far as I know them, is, that apparently in treating average- and low-grade ore of the district amalgamation can be eliminated; that if amalgamation is eliminated and solution introduced at the stamps, a considerable saving in operating cost, cyanide, and dissolved gold losses is possible.

Unquestionably more extensive study has been given to the treatment of the Dome ore than to any other in the district. Hence one hesitates to make what may appear to be a criticism of an operating system probably justified by a careful balancing of co-ordinate factors by an eminent firm of metallurgists. However, no metallurgical discussion would be complete without touching upon this point, which is the salient difference between the two metallurgical systems of the district.

Concentration versus Non-concentration.—The Hollinger is the only mill making a table concentration. About 16 per cent. of the gold in the ore is recovered in the concentrate, and the advantages claimed would indicate that the possibilities justify careful consideration. The pulp, with the concentrate removed, needs much less careful treatment than the entire pulp, concentrate included, would require, hence a small tonnage of concentrate may be given whatever treatment it demands to get the best result, while the large tonnage of pulp free from concentrate may receive the much smaller amount of attention it requires.

Table concentration at the Hollinger costs about 5c. per ton of ore treated and recovers about 80 lb. of concentrate per ton of ore, assaying about $2\frac{1}{2}$ c. per lb., worth, therefore, about \$2. In the careful treatment given the concentrate the value per pound of concentrate is brought down to about 0.3c.; in other words, a saving of \$1.76 is made from the 80 lb.

of concentrate from each ton of ore, which is a large enough amount to warrant the expenditure of 5c. to safeguard. The performance of the thickeners and filters is improved if a feed can be maintained composed of particles of one specific gravity, so that from the standpoint of better mill performance, due to keeping the concentrates out of the thickeners and filter, I am of the opinion that the expenditure of 5c. per ton of ore is justified where the concentrates taken out amount to say 4 per cent. or more of the total tonnage. Another advantage is that even after a very careful treatment the concentrate tailing assays from \$5 to \$7 per ton of concentrate, equal to 12c. to 20c. per ton of ore, which at some later date may be saved if concentrate is segregated and impounded. Also the saving in treatment cost, due to the less agitation required for the pulp freed from concentrate, should be credited to concentration. Hence, I should say that if table concentration cost 20c. per ton, instead of 5c., it would still be justified when a considerable amount of Class B ore is to be treated.

Agitation.—The ore particles composing the pulp coming to the agitators are extremely quick-settling and largely granular; after only a few minutes' shutdown the pulp compacts solidly in the bottom of the agitator so that, mechanically, agitation is a difficult problem. At the Dome four Pachucas in series are in operation, but all the other mills use the Dorr agitator, which seems to be peculiarly adapted to the local requirements. From the trouble experienced in keeping the Hollinger pulp in suspension in the filter-loading vats I judge that the Pachuca agitator is expensive in power required to prevent the filling up of the cone. With a trifling amount of power and a normal air consumption the Dorr agitator meets all the mechanical difficulties.

Metallurgically the quicker-settling particles need longer treatment than the lighter material. Selective agitation of the quick-settling particles is therefore essential if the best results are to be obtained.¹ The Dorr agitator, allowing as it does control over the rate of flow through the tank of material of greater or less than the average settling rate, meets the metallurgical requirements of the ore very nicely.

Filter Methods of the District Compared.—The Merrill filter is in use at the Dome, and while direct treatment in the presses has been tried, it has been found that the use of agitators for the dissolution of the precious metals is preferable, as a very large and expensive filter installation would otherwise be required. The Moore filter at the Hollinger has indicated that the ore contains such a large proportion of quick-settling material that vacuum filtration is not altogether satisfactory. In the loading vats, six air lifts are used, requiring 40 h.p. at the compressor, and even with this intense circulation the heavy slime accumulates on the 60° hopper bottoms, resulting in such damage to the leaves that about one-

¹ For an exposition of the term "selective agitation," see the paper by J. V. N. Dorr, in *Bulletin* No. 92, p. 2072 (August, 1914).

third of the filter-operating cost is represented in repairs to filter leaves. Then, too, with the strong circulation, due to the air lifts, the cakes are channeled and uneven. On the whole, I do not consider that vacuum filtration is adapted to the Porcupine ore.

At the McIntyre Mill, a Burt filter is doing very good work and is stated to have been entirely satisfactory. At the Hollinger and the McIntyre mills, the pulp from the new sections will be treated by continuous counter-current decantation, while the pulp from the original sections will continue to be put through the filters, so that shortly some interesting comparative figures on the two methods should be available.

Continuous Counter-current Decantation.—It has been previously mentioned that no oxidized ore occurs in the district, and the clean undecomposed rock breaks down to give an ideal product in the thickeners. Class A ore makes no colloid, and Class B ore, while grinding to an extremely fine, amorphous product, gives little trouble in settling, owing to its high specific gravity. Class A ore can be thickened to 30 per cent. moisture and Class B to 35 to 45 per cent., depending upon the percentage of concentrate. The critical moisture is 45 per cent., when 5 per cent. of concentrate is present, and about 35 per cent. moisture with Class B pulp, free from concentrate. On account of being able to get such unusually low moistures in the thickeners, a very high recovery of dissolved metals is possible by continuous decantation. Also, the fact that the cyanide strength of the solution from agitators to thickeners need only be carried at slightly above 1 lb. per ton favors the decantation system, where the mechanical loss of cyanide is generally higher than in ordinary filter practice. At the Porcupine Crown, with about \$13 going into solution per ton of ore and using four steps of decantation, with no filter, the dissolved gold loss is only 5c. and the mechanical loss of cyanide only 0.32 lb. per ton of ore.

The Hollinger mill put in two steps of continuous decantation early in 1913, and the complete counter-current decantation system was installed in the cyanide extension of the Porcupine Crown later the same year. Later still, the Vipond installed the counter-current decantation system, as did the McIntyre when the mill was enlarged. The Hollinger has a complete 300-ton plant under construction and the Acme mill a 600-ton plant, both to use this system.

Mill Design.—The proper design for a mill treating Porcupine ore will depend upon the proportions of Class A and Class B ore to be handled. Unless there is to be a large excess of Class A ore, amalgamation may be dispensed with, as the recovery by amalgamation will not warrant its use. If Class A ore is in large excess it would still be an open question, but from a recovery standpoint amalgamation is unnecessary.

The Hardinge ball mill may not show up as well on Class A as on Class B ore, but I am inclined to think that it would. With an excess of Class

B ore the ball mill will be superior to stamps. I am of the opinion that a cylindrical tube mill should be used for fine grinding, rather than a conical mill, if only for a theoretically better dissolution of coarse gold.

For the treatment of any considerable proportion of Class B ore, table concentration, with separate treatment of the concentrates, will probably pay.

Agitation should be arranged to be continuous, preferably in a series of flat-bottomed agitators, allowing a preferential treatment for the quicker-settling portion of the ore.

If filtration is used, a pressure filter will be more satisfactory than a vacuum filter; however, the ore is so perfectly adapted to continuous counter-current decantation that this would seem to be the proper treatment.

On account of the severe winter conditions and the high cost of fuel, the object to strive for in the design should be as compact an arrangement of the equipment as possible, so as to minimize the cubic area of buildings to be heated.

In the district, the water supply is ample, the sites for mills are good, and the facilities for convenient tailing disposal are adequate.

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Gold-Bearing Gravels of Beauce County, Quebec.

BY J. B. TYRRELL, TORONTO, ONT., CANADA.

(New York Meeting, February, 1915)

A SHORT time ago I paid a visit to the alluvial gold fields on the tributaries of the Chaudière River in Beauce County, Quebec, in company with A. O. Dufresne, late manager of the Champs d'Or Rigaud-Veaudreuil, and now Assistant to the Superintendent of Mines of the Province of Quebec. As the conditions under which the gold occurs in this district are not very generally known, and present some interesting features, a brief description of these conditions, and a consideration of the causes which gave rise to them, may be of interest to other mining engineers.

During the latter half of last century the country was visited by many mining engineers and geologists, and many references to it may be found in the reports of the Geological Survey of Canada between 1848 and 1911. The most important of these are by J. A. Dresser and J. Keele and the late Robert Chalmers. The Department of Colonization and Mines of the Province of Quebec also published a report with map on the district by J. Obalski.

From the earliest times the valley of the Chaudière River formed one of the main avenues of approach to the St. Lawrence in the vicinity of Quebec from the country to the south as far as the seaboard of the States of Maine and Massachusetts. The Indians had a well-known trail along the banks of the stream, and armed troops and foraging parties constantly moved backward and forward along it between Quebec and New England in those insecure times before the ceding of Canada to Great Britain.

Even yet the natural advantages of this old military route are recognized, and the government of the Province of Quebec has now under construction along it a magnificent highway from Quebec to the International Boundary Line, which is to form part of a through highway from that city to Boston.

The land along the valley is naturally well suited for agriculture, and about the end of the eighteenth and the beginning of the nineteenth centuries farmers began to go back from the St. Lawrence, and to occupy the higher lands along the upper portions of the Chaudière Valley. This settlement has gone on until now the country is peopled with a prosperous agricultural population.

DISCOVERY OF GOLD

In 1823 or 1824, a woman first discovered gold in the Chaudière Valley near the mouth of Gilbert River. No attention was paid to the discovery, but in 1834 a young girl named Clothilde Gilbert, taking a horse to water, found in the creek, close to the location of the previous discovery, a nugget of gold weighing 44 dwt. Eleven years later the DeLery family, owners of the seigniory of Rigaud-Vaudreuil, obtained a patent from the Crown giving them exclusive privileges forever to mine the precious metals within their seigniory.

In 1847, the year before gold was discovered in California, the Chaudière Mining Co., which leased the mining rights from Mr. DeLery, mined gold on the Gilbert and Des Plantes rivers, and during the three following years continued to operate on the Gilbert River.

In 1851, the mining rights of the whole seigniory were leased to Dr. James Douglas and others of Quebec, who continued operations, chiefly on the Gilbert River, until 1864. After this date mining was prosecuted with more or less activity for about 30 years.

In all, up to the end of the century, about \$2,000,000 worth of gold was extracted from the gravels of the Gilbert River valley, while it would seem that about \$500,000 worth of gold was extracted from the gravel of the other tributaries of the Chaudière River.

CHARACTER OF COUNTRY

That portion of the watershed of the Chaudière River and its tributaries, from whose buried gravels gold to the value of \$2,500,000 has been extracted, extends for 20 miles in the direction of the valley, and 6 miles transverse to it, forming a block of land about 120 square miles in area, in which placer mining has been more or less systematically prosecuted. It lies in Beauce County, Quebec, 50 miles southeast of the city of Quebec, and 25 miles west of the International Boundary Line between Quebec and the State of Maine. The principal town is Beauceville, with 1,700 inhabitants, situated on both banks of the river at an elevation of 500 ft. above the sea, with hills rising to heights of 600 or 700 ft. both to the northeast and southwest of it. Transportation to or from the district is afforded by the Quebec Central Railway, which at the present time runs two passenger trains a day each way to and from Quebec. The railway runs up the valley of the Chaudière River through a number of small prosperous towns which are located on the bank of the stream, while back from the river the country is laid out in farms which are for the most part cleared of timber and in a good state of cultivation. Two wagon roads run up the valley, one on each side of the stream, and the method which has been generally adopted here, as elsewhere in Quebec, of surveying farms with a

narrow frontage on the river and a long extension back from it, permits the farmers to live moderately close to one another beside the main roads, giving these roads the appearance of long-extended scattered villages.

The country in which the gold-bearing district is situated is a dissected plain or tableland with a mean elevation of 1,000 or 1,100 ft. above the sea, lying between two old and greatly degraded mountain ranges.

These mountains are the northern extensions of the Green Mountains of Vermont and the White Mountains of New Hampshire. They run in two parallel chains about 50 miles apart northeastward from the International Boundary into the Gaspé Peninsula. The stronger chain, which has been called the Megantic Range, runs along the International Boundary Line, and some of its peaks rise to heights of 2,500 or 3,000 ft. above the sea. Some of the peaks of the other chain, known as the Sutton Range, rise as high as those farther to the southeast, but taken as a whole this range is the lower of the two.

Between these two ranges of mountains lies an extensive tableland which has been worn down by long-continued atmospheric erosion into rounded hills and wide valleys. The summits of the hills are covered with a thin mantle of glacial drift, while the lower slopes are rounded up by a thicker layer of the same unassorted material. In their native condition the hills were completely covered by magnificent forests of pine and maple, now largely cut down since the land has been brought under cultivation.

DRAINAGE

The general direction of the drainage from this tableland is either northeastward or southwestward, parallel to the mountains. Nevertheless, it is trenched across, and the Sutton Mountains are cut through by the great transverse valley of the Chaudière, which collects the water from the many normal longitudinal streams, and carries it down into the St. Lawrence River. This valley has been cut deep into the old plateau and has reached a fairly mature condition, with gentle slopes descending from the high lands on both sides to the river, which has a moderate and fairly regular grade of about 8 ft. to the mile from the upper portion of the area under consideration to the St. Lawrence River. Such minor obstructions as do occur in the stream, as at the Devils' Rapids, have probably been caused by diversion of the river from its old channel by glacial agencies.

STRUCTURAL GEOLOGY

The rocks that compose the Sutton and Megantic mountains are pre-Cambrian gneisses, and talcose, chloritic, and micaceous schists.

Between these mountain ranges, in the region of the Chaudière, the plateau country is underlain by green and reddish slates, quartzites, and

sandstones, which are stated by the officers of the Geological Survey of Canada to be of Cambrian and Cambro-Silurian age. In many cases these slates, etc., present a remarkable similarity to the pre-Cambrian slates and schists of Keewatin age in northern and western Ontario. Some of the slates are ordinary water-worn sediments, while others have recently been proved to be ash rocks, or similar rocks of igneous origin.

These rocks were deposited in a horizontal attitude in the seas of the Paleozoic era, but have been squeezed and crushed so that they are now generally steeply inclined or even vertical, and strike about N. 45° E., parallel with the mountain ranges.

Through the schists and slates, dikes and bosses of igneous rock, varying in character from peridotite to quartz-porphyry, have been injected. It is highly probable that some of the igneous rocks intercalated with the slates were injected into them as sills or laccolites before they were tilted and folded into their present attitude, but some of the dikes are doubtless subsequent to the folding. However, it is significant of the age of the igneous rocks associated with the gold-bearing gravels in the vicinity of Beauceville, that some of the green schists, associated with and included in the folding of the Cambro-Silurian rock in the valleys of Mill Creek and Chaudière River, were found to be volcanic rhyolite tuffs, while the igneous rocks in the vicinity are quartz-porphyrines of similar composition, and probably of somewhat similar age.

Mr. Dresser¹ has already pointed out the similarity in character of these intrusives to those of the Stoke Mountain range farther west, which are associated with some of the most important copper properties in Quebec.

In the valley of Gilbert River, from which most gold has been collected, quartz-porphyrines and acid intrusives, either sills or dikes, are particularly abundant. In the vicinity of many of the more acid intrusives quartz veins have been found to occur containing more or less gold associated with such sulphides as pyrite, chalcopyrite, and galena. These are all the hard rocks known to exist in the district under consideration, and such sediments as overlie them consist of unconsolidated material of very much younger age.

The oldest of the later sediments consist of thin beds of stratified gravel extending down the bottoms of the valleys, but of no considerable lateral extent. In places they contain grains and nuggets of gold. Overlying these gravels is a varying thickness, sometimes as much as 100 ft., of unassorted and unstratified boulder clay. Other and later sands and gravels also occur in gorges in the bottoms of the valleys, which also contain a small quantity of gold. Overlying these is a second thickness of boulder clay. Finally there is gravel in the bottoms of the present streams.

¹ J. A. Dresser: The Bedrock of the Gilbert River Gold-fields, Quebec. *Journal of the Canadian Mining Institute*, vol. viii, pp. 259 to 262 (1905).

Historical Geology

The sequence of events which led up to the formation of these buried gold-bearing gravel deposits was about as follows:

After both the igneous and sedimentary rocks of early Paleozoic and pre-Paleozoic times had been formed or deposited and had been intensely crushed and folded into what must have been a range of mountains, they appear to have been intruded by dikes of the following igneous rocks: Peridotite, pyroxenite, gabbro and diabase, granite, quartz-porphyry, etc.

Subsequent to these intrusions, probably to the last of them, the rocks were again subjected to heavy strains, so that they were still further fractured. Into some of the more acid of the igneous rocks (whether dikes or sills is not always certain), siliceous waters carrying sulphides of iron and copper, with native gold, were introduced along the fractures, also from these fractures the gold-bearing solutions seeped out into the adjoining rocks, forming quartz veins and pyritized zones carrying a smaller or larger percentage of gold. Thus the veins were formed from which the grains and nuggets of gold found in the valley gravels have undoubtedly been derived.

Toward the close of the Paleozoic era, and after the rocks had assumed a fairly stable condition, the whole country was raised above the level of the sea, and since that time it would appear to have remained above sea level, and to have been exposed constantly to the influence of atmospheric and stream erosion and denudation. During this vast period of time, extending from the end of the Paleozoic era to the present, an enormous thickness of rock was undoubtedly removed from the general surface, and as the softer rocks would be worn away faster than the harder ones, the latter remained as higher points and ridges.

At first the water which drained from the district would flow downward to the sea over the lowest parts of the surface, irrespective of the hardness of the rocks of which this surface was composed, and water courses so begun might persist to the present. The great valley of the Chaudière is probably such a persistent water course, while the smaller streams have been cut off from their direct connection with the sea, and have been obliged to become tributary to the Chaudière, their courses being finally determined by the varying characters of the underlying rock.

While the surface was being decomposed through the agencies of air and moisture, with the help of plants and animals, the decomposed rock was constantly being carried downward by the rills and streams, and at the same time was being assorted into heavier and lighter portions. In this process the coarser and heavier portions constantly lagged behind and became entrapped by the inequalities of the underlying rock, while the smaller and lighter portions were carried down into the main channel of the Chaudière River, and thence into the sea.

In this way, during the long period which intervened between the up-

lift near the close of the Paleozoic era and the beginning of the Pleistocene period, the country was worn down, possibly from a high range of mountains, to a fairly mature physiographic relief, in which rocky cliffs and gorges were unknown, and the slopes of the hills were everywhere gentle, with coverings of decomposed residual rock. Also in the bottoms of the wide valleys the streams flowed with gentle regular current without rapids or waterfalls. In and beside these streams were deposits of sand and gravel which undoubtedly contained most of the heavy minerals that had been washed down from the adjoining hills during the whole period of their long-continued erosion, unless these minerals had been carried away in solution, or were in a sufficiently fine state of division to have been transported to the sea with the lighter sediments. Of these heavy minerals the most important, and at the same time the most persistent, was gold.

The general relief of the country at the beginning of the Pleistocene period would have been very much like that of the Klondike district, in Yukon Territory, at present, particularly those parts of the Klondike drained by Dominion Creek and Indian River, where later gorges have not been developed; with this difference, that the Quebec slopes were easier and the whole topography was more mature.

Another point of similarity between the two districts is, that throughout the whole time when active erosion was in progress the drainage of the country was local and the whole of the gravel concentrated in the bottom of any valley was derived from that particular valley or its tributaries, and not from a foreign valley.

Again, the gravel in the bottom of a valley was the ultimate concentrate from the vast quantity of material which had been eroded from that valley, possibly aggregating many cubic miles of rock, and consequently if the gravel was rich in gold it was due to the quantity of rock concentrated, rather than to the original high gold tenor of the rock.²

At the beginning of the Pleistocene period there was a break in the continuous course of atmospheric and stream erosion which had been in progress throughout the Mesozoic and Tertiary epochs, for snow and ice began to collect in great quantity on the Adirondack Mountains to the south, and from this center or gathering ground the ice moved northward down the valley of the Chaudière River, across the hills which flank it on both sides, and over the valleys of the tributaries which flow into it approximately at right angles to its course, until it stopped in the vicinity of the south bank of the St. Lawrence River.³

² Cf. The Gold of the Klondike, by J. B. Tyrrell, *Transactions of the Royal Society of Canada*, vol. vi, New Series, Sect. 4, pp. 29 to 59 (1912), and The Law of the Pay-streak in Placer Deposits, by J. B. Tyrrell, *Transactions of the Institution of Mining and Metallurgy*, vol. xxi, pp. 593 to 613 (1912).

³ For detailed evidence of the succession of events during the Glacial Period the reader is referred to Robert Chalmers's Report on the Surface Geology and Auriferous Deposits of South-Eastern Quebec, *Annual Report of the Geological Survey of Canada*, vol. x, New Series (1897).

From the standpoint of the miner engaged in the exploitation of alluvial gold-bearing deposits, this first ice invasion from the south is of great interest, for, inasmuch as it moved down the Chaudière Valley, where this valley runs northwestward, it doubtless removed any stratified sand and gravel which may have been in the bottom of those portions of the valley so oriented, and at the same time it rounded up the sides of the valley, and filled in the mouths of the lateral valleys with débris collected from the valley itself or from the sides of the adjoining hills. During its later waning stages it probably also left lateral moraines on both sides of the valley.

When at its greatest extent, this Adirondack glacier covered the higher lands and moved over the valleys of the tributaries of the Chaudière River which were transverse to its general course. In these cases it moved the decomposed rock from the summits and the south sides of the ridges down into the valleys and covered the gravel, which had previously been deposited there, with a coating of boulder clay or till.

In most cases, as in the valley of the Gilbert River, the glacier had lost the greater part of its pushing power when it reached the lower levels, so that it left the gravels undisturbed and merely covered them with its heavy coating of dirt brought from above. In some few cases, as in some places on the banks of Meules Creek, there was still a little vertical energy left in the glacier when it reached the bottom of the valley, and so it kneaded up the sand and gravel into a compact unstratified mass of water-worn material a few feet in thickness before covering it with unassorted till.

While this northwestward moving glacier pushed a certain quantity of loose unassorted material into these smaller transverse valleys it did not fill them, but deposited its load on their southern slopes, and consequently when it retired it left the new bottoms of these valleys farther north than they were before, while the old pre-glacial gravels in the original bottoms of the valleys were buried under the talus of rock débris to the south.

When the Adirondack ice withdrew from the country at the close of the first glacial period, the brooks and rivers flowed in the same valleys which they had occupied before the ice invasion, but as the bottoms of the transverse valleys had been moved toward the northwest the streams naturally adopted the lowest parts of the valleys, and therefore now flowed in channels northwest of their former channels, and usually at somewhat higher elevation; at the same time they were cut off from the main Chaudière Valley by the ridges or lateral moraines which had been piled up along its sides. Consequently, in their endeavor to reach the main stream, the lateral brooks cut new gorges in the bottoms of the valleys northwest of the old channels, but their sides remained steep, for the period during which the country was free from ice does not appear to have been

sufficiently long to have permitted of the grading of the sides of these second gorges to gradual slopes. One of these interglacial gorges has been outlined by shafts and drill holes on the northwest side of Meules Creek.

After the deep, narrow, interglacial gorges had been formed the country was again, and probably more deeply, covered with ice, but on this occasion the ice accumulated on the Laurentian hills north of the St. Lawrence and then moved southward and southeastward across the St. Lawrence River and up the long slope south of it for about 100 miles almost to the summit of the Megantic range of mountains on the International Boundary Line. This second invasion of ice therefore moved up the valley of the Chaudière River in the opposite direction to that in which it had moved on the former occasion. Again it scored out and smoothed off the bottom and sides of the main valley. Also, as it passed over the valleys tributary to the main valley, and at right angles to its course, it pushed such decomposed and broken rock as it was able to collect down into these valleys, covering their northern sides with *débris* and filling

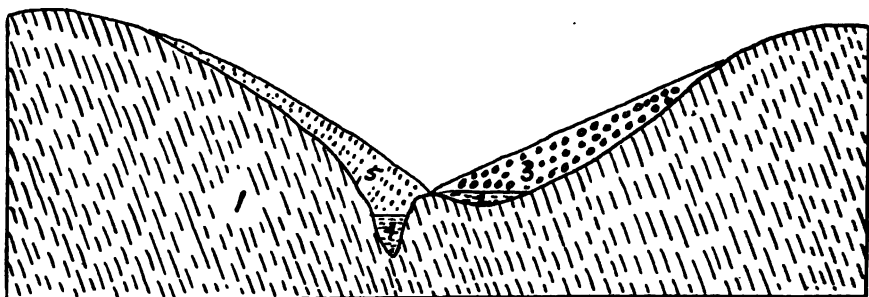


FIG. 1.—DIAGRAMMATIC SECTION ACROSS THE VALLEY OF MEULES CREEK, WHICH FLOWS NORTHEASTWARD INTO THE CHAUDIÈRE RIVER.

1. Paleozoic slate.
2. Pre-glacial gold-bearing gravel.
3. Boulder clay of the Adirondack glacier from the southeast.
4. Interglacial sand and gravel.
5. Boulder clay of the Laurentian glacier from the northwest.

in and covering up the interglacial gorges which had recently been cut in them, but it did not completely fill the valleys with boulder clay, so that when this glacier in its turn melted away and disappeared, and open streams again began to drain the country, they flowed in channels independent of either of the earlier channels, and in some cases at least, intermediate between them. (See Fig. 1.)

Since the close of the last glacial period, when the ice finally retired from the country and left it in much the same condition as it is at the present time, the streams in the transverse valleys are again cutting new channels for themselves in the bottoms of the valleys through the covering of boulder clay and down into the underlying rock on lines independent of the earlier channels.

A striking feature of the new system of drainage which prevails in the country at the present time is that the lateral streams discharge into the main valley of the Chaudière over rapids or waterfalls from "hanging valleys." This condition indicates clearly that the lower parts of these lateral streams are not now occupying their old pre-glacial or interglacial channels. In no case was I able to learn of either one or the other of the old channels having been traced all the way down into the Chaudière channel.

When the ice had finally retired it left the whole country, both hills and valleys, covered with a sheet of glacial drift. On the hills this sheet is usually thin, while in some parts of the valleys it may reach a thickness of 100 ft.

This sketch of the causes which led to the formation of the beds of gold-bearing alluvial gravels, and of the methods which Nature adopted in giving them their present characteristics, and in hiding them in their present obscure locations, may be summarized as follows:

Summary of Gold Conditions

1. Gold was probably introduced into the folded Paleozoic rocks subsequent to, but in close association with, sills or dikes of acid rocks, such as quartz or granite-porphyry.

2. It was introduced along with pyrite and other sulphides in siliceous water which formed quartz veins in or near these dikes, etc.

3. Toward the close of the Paleozoic Era the country was raised above the level of the sea, and has remained above the sea until the present time.

4. Throughout the most of this immensely long period, until the beginning of the Pleistocene period, it was constantly suffering erosion from atmospheric and stream agencies, and it was worn down to a fairly mature condition with gently sloping hills and wide valleys.

5. Where gold occurred in these hills it had been washed down through countless ages into the bottoms of the valleys, and was concentrated in the alluvial gravels beneath and beside the streams.

Many streams throughout northern Canada which flowed over gold-bearing rocks must also have had gold-bearing gravel in their beds in pre-glacial times. In most cases, however, the subsequent glaciation was sufficiently severe to have carried away all this gravel, while in the Chaudière district the glaciation was less severe, and some of the gravel was left in place.

6. After this long period of erosion and concentration a great glacier formed on the summit of the Adirondack Mountains and moved north-westward over the country toward the St. Lawrence River. On its way it crossed the valleys which lay transverse to its course, and buried some of the gravel which lay in the bottoms of those valleys under a heavy

mantle of boulder clay. Sometimes the gravel was left quite undisturbed in its original condition, sometimes it was kneaded together so that its stratified character was obliterated. It is chiefly from these pre-glacial beds of gravel that gold has been extracted.

As the glacier moved directly down the Chaudière valley it probably scored out most, if not all, of the gravel which had accumulated in it, though up to the present time this question does not appear to have been definitely settled, for the bottom of the valley has not been thoroughly prospected either by shafts or drill holes. At one place, namely at Devils' Rapids, gold has been found in the Chaudière River, but here the stream is flowing for a short distance transverse to the general direction of the valley and the course of glaciation.

7. After the Adirondack glacier had retired, new and narrow channels were cut by the transverse streams in the bottoms of the transverse valleys, to the north of the old pre-glacial channels. These contain a small quantity of gold, but the interglacial period was not sufficiently long to permit of the concentration of much gold in them, so that except where they may possibly have cut into or across the earlier pre-glacial channels they have not proved, and are not likely to prove, rich in gold content.

Up to the present time interglacial channels do not appear to have been distinguished from pre-glacial ones, and doubtless some of the failures which have occurred in the district have been caused by expending time and energy on the buried, but poor, interglacial channels, under the impression that they were the rich pre-glacial channels.

8. After the interglacial channels were formed another glacier advanced across the country from the northwest and buried these channels under another and later sheet of boulder clay.

9. When this glacier retired from the country the streams began to cut out their present channels, which are independent of the two former sets, but as yet no large quantity of gold has been concentrated into these new channels.

Whether all the buried gold-bearing gravels have been discovered or not will not be discussed here, but it may be pointed out that the pre-glacial channels of the lateral streams, which in their upper courses are gold-bearing, do not appear in a single case to have been traced down to their junctions with the main valley of the Chaudière River, though it is reasonably certain, from the mature character of the topography throughout the country, that such channels are continuous without falls or interruptions from the lateral valleys into the main valley.

CHARACTER OF BEDROCK

The bedrock underlying the pre-glacial gold-bearing gravels consists chiefly of green and gray chloritic and quartzitic slates striking N. 45° E.

and dipping southeastward at an angle of 70° or steeper. Where these slates are overlain by pre-glacial gravels they are rough and uneven and form excellent natural riffles, so that the gold was collected either in the inequalities of their surface, or immediately above them, and they have not been smoothed and polished by glacial agencies like the rocks of the adjoining hills.

On the Gilbert River and in other localities these slates were intruded by sills or dikes of quartz-porphyry, and these sills or dikes occur at places which are said to have been the most productive in the whole area; but in the absence of personal observation of the old mines it is impossible for me to say what effect the character of this bedrock had on the pay streak in the old channels.

CHARACTER OF GOLD

The gold obtained from the gravels of the tributaries of the Chaudière River is mostly such as is usually known to placer miners as coarse gold, very little of it being in the form of very minute flakes or particles. There can be little doubt but that fine gold existed in the veins from which the placer gold was originally derived, but if it did it was carried farther down the streams and much of it was probably deposited in the gravels of the Chaudière River. One nugget was found on the Gilbert River which weighed 51 oz., 18 dwt., 6 grains, another weighed 45 oz., 12 dwt., while another nugget found on the same river weighed 42 oz.

Last summer Louis Matthieu recovered 50 oz. of gold from the gravels of Meules Creek, all of which was quite coarse and granular. The largest nugget weighed between 2 and 3 oz., while the next largest, which I obtained, weighed 24 dwt., 12 grains.

Mr. Obalski gives the fineness of two samples of gold as 874 and 879, equal to a value of \$18.06 to \$18.15, and these may be considered as representing the average fineness of the gold of the district.

METHODS OF MINING

In the earliest days of mining in the district the gravel was collected from the bars in the river, probably where the river crossed or cut into one of the old channels, and was washed in a gold pan or in a cradle or rocker to recover the gold.

Afterward some parts of the pre-glacial channels were discovered which were covered with but thin layers of boulder clay. The boulder clay was thrown to one side, and the underlying gravel was shoveled from the open pits into sluice boxes, supplied with water from higher up the river, and the gold was collected in the boxes.

At a later period the rich gravels were found under a heavy over-

burden of boulder clay, in places 60 or 70 ft. thick. In some cases this buried gravel was reached by tunnels driven into the sides of the hills, and in other cases by vertical shafts sunk to it. From the ends of the tunnels and from the bottoms of the shafts as much of the gravel and underlying bedrock as contained gold was mined and brought to the surface, where it was washed in sluice boxes as before, and the gold extracted.

Four or five years ago a much more ambitious plant was installed on Meules Creek. A ditch 7 miles long was dug from Lake Fortin, at the head of Mill Creek, in which water was conducted to a penstock on the high ground south of Meules Creek. Thence it was conducted in a pipe to a point on Meules Creek where hydraulic operations had been determined upon, the head of the water at this point being 260 ft. Here one or more hydraulic giants were installed and through them the water was projected against the south side of the valley, and the gravel and débris were washed down and run through a sluice to collect the gold. At the tail of the sluice a bucket elevator picked up the tailings and stacked them lower down the valley. Unfortunately, the operations do not appear to have been financially successful, for the bank which it was proposed to wash down proved to consist of boulder clay with but a thin layer of re-assorted pre-glacial material beneath it, which was not sufficiently rich to compensate for the poverty of the boulder clay above. The plant has not been in operation for the past two summers.

During the past summer a few tributers or laymen were working in a small way "shoveling in" on Meules Creek, with the result stated at the beginning of this paper, but mining work appears to have ceased on Gilbert, Des Plantes, and other streams in the vicinity some years ago.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Oil and Gas Possibilities of Kentucky

BY F. JULIUS FOHS, TULSA, OKLA.

(New York Meeting, February, 1915)

WITH portions of two coal basins within its borders and a few scattered fields already developed, the question arises: What is the future of Kentucky as an oil-producing State? Is the long list of failures due to lack of commercial pools, or unintelligent prospecting? A study of its beds and irregularities of structure points not only to a large waste of development money on unpromising areas, but also to the presence of a few structures well worthy of development.

The surface rocks of Kentucky show in succession more than 4,000 ft. of Paleozoic sediments and more than 2,000 ft. of Cretaceous, Tertiary, and more recent deposits. Folding and erosion have brought these beds to the surface, where they have been observed and studied in detail by my associate, James H. Gardner, myself, and others. This has given opportunity to observe the beds offering suitable reservoirs and having the proper impermeable covering; also the mapping of outcrops and outcrop lines, taken in conjunction with available well records, indicates that certain areas are worth testing, and with even greater definiteness shows areas which should be excluded, as devoid of possibilities.

West of the Tennessee River, Cretaceous, Tertiary, and Quaternary sediments occur so that all trace of ancient folding is obliterated, the sediments overlapping unconformably Mississippian rocks. This area embraces 2,000 square miles, or one-twentieth of the area of the State, the Paleozoic rocks covering the remainder. Since there is nothing upon which to base the location of tests for oil and gas in the Cretaceous-Tertiary beds of Kentucky, the Paleozoic area will be chiefly considered.

The distribution of Paleozoic rocks in Kentucky is centered about the north-northeast striking Cincinnati geanticline, bringing to the surface on the Jessamine dome the oldest rocks exposed in the State, those of the Devonian, Silurian, and Ordovician systems. On either flank of this great earth-arch are the Mississippian rocks, sloping gradually beneath the coal-measure basins to the west and to the east. West of the western coal basin, and between it and the Cretaceous-Tertiary rocks further west, is a high area of Mississippian rocks. Crossing the State

in an east-west direction is the Chestnut Ridge anticline (a disturbance recently shown by Mr. Gardner to extend from the Ozarks to the Appalachians),¹ consisting in Kentucky of the Rough Creek uplift, the Kentucky River fault zone, and the Warfield anticline. The Jessamine dome of the Cincinnati arch is due to the crossing of the Chestnut Ridge disturbance.

Beginning at the west, between the Tennessee and Tradewater rivers, lies the much-faulted area constituting a portion of the fluorspar field of western Kentucky and southern Illinois. Its structure results from the crossing of two monoclinal folds causing a fan-like fold spread with maximum dip to the southwest and the maximum uplift beyond the Tradewater River, northeast in the Rough Creek uplift of which it forms a part. Faulted with displacements exceeding 1,500 ft., and with heavy shale beds absent, presenting every opportunity for escape of oil and gas, search for them in this section would be useless. This embraces most of Crittenden, Livingston, Caldwell, Trigg, Lyon, and Christian counties. In the more southerly portion of these counties, toward the Tennessee State line, gentler dips and virtually no displacements occur, but all trace of oil structure is obscured by the red clay and chert of the Mississippian limestones, making this portion of the section one of uncertain availability.

Considering next the western coal field, we have a basin area split in an east-west direction by the Rough Creek uplift, which has a fault zone paralleling its northern slope. This uplift exposes a belt of Chester or upper Mississippian limestone and sandstone, while on its southern slope small domes occur, due to the crossing of north-south running anticlines. These domes are favorable for oil, and on one of them Mr. Gardner located the Ohio County pool. Some small exposures of oil formations have been found north of the uplift, but more favorable opportunities occur in the basin on the south. On the extreme south side of the basin the beds outcrop. The LaSalle anticline of Illinois dies out before reaching this coal basin in Kentucky. The districts in the vicinity of the asphalt-rock occurrences in Edmonson County, and of certain other localities in this basin, are favorable for further investigation, and some may warrant testing.

Locally small domes may occur on or in the vicinity of the Rough Creek uplift which would warrant testing, but such areas are few in number and restricted in size, and must be selected with great care.

On the south side of the uplift a syncline, named the Moorman by Hutchison, occurs. Its north slope is broken by faults and most of the contours on the No. 9 coal are low; probably too low to have trapped oil. On the south slope, the contours rise higher and an occasional dome or half dome occurs, usually though not always faulted. With 75 percent.

¹James H. Gardner; Extension of Chestnut Ridge Anticline. *Bulletin of the Geological Society of America* (1914).

of the beds consisting of shale, it is probable that many of these faults are sealed. By selection of the more favorable elevations, giving due attention to the position of the faults and their effects on the accumulation, it is possible that tests might prove the presence of a number of small pools. The south border of the coal field is a series of *en echelon* faults. Within the field and paralleling these along the line of an old fold, roughly paralleling also the main uplift, is a complex series of *en echelon* fault blocks, and testing for oil in Chester sands south of these blocks appears likely to prove of small consequence.

The Yelvington anticline, near Maceo in Daviess County, has recently been tested, but whether the hole was so placed as to be a fair test I am unable to say.

North of the Rough Creek uplift and east of the western coal field, in Meade and Breckinridge counties, minor folding and doming occur with Mississippian rocks, chiefly Chester rocks at the surface; here, in addition to small gas wells already developed, a few favorable localities are known, which will probably yield both oil and gas.

Bordering the western coal field on the south and east other gentle folds occur, notably in Logan and other counties, and this area well deserves testing. Here also Chester rocks occur at the surface. It is noticeable that in many instances, where doming occurs on the border of this coal field, Chester sands proper, when they occur at the surface, are bituminous, clear evidence of accumulation of oil prior to the erosion which has since laid them bare and permitted the evaporation of most of the volatile constituents.

The area lying within the limits of the main Cincinnati uplift is partly faulted and lacks the necessary structural relief; in addition, the Niagaran and Corniferous sands are either eroded, or occur at very shallow depths; it offers very limited possibilities.

In and adjacent to the eastern coal field a number of undeveloped pools probably occur. In Whitley and Knox counties, immediately east of Wayne County, owing to the proximity of the Pine Mountain uplift to the east and the absence of marked secondary folds, other than the main syncline between this uplift and the crest of the Cincinnati geanticline (only 70 miles distant), good-sized pools are not to be expected. Shortly west of this main syncline are the small synclinal folds, which control the production in Wayne County, some of which remain undeveloped. Farther northeast, the axes of the Cincinnati and Pine Mountain uplifts diverge and two prominent east-northeast folds, the Sandy Hook and Warfield anticlines, appear. The igneous dikes of Elliott County occur near the Sandy Hook fold and are probably related to it. On the flanks of both folds, where terracing or doming occurs, pools may be expected, especially on the south flank of the Warfield fold in portions of Knott, Perry, Clay, Laurel, Leslie, Breathitt, Floyd and

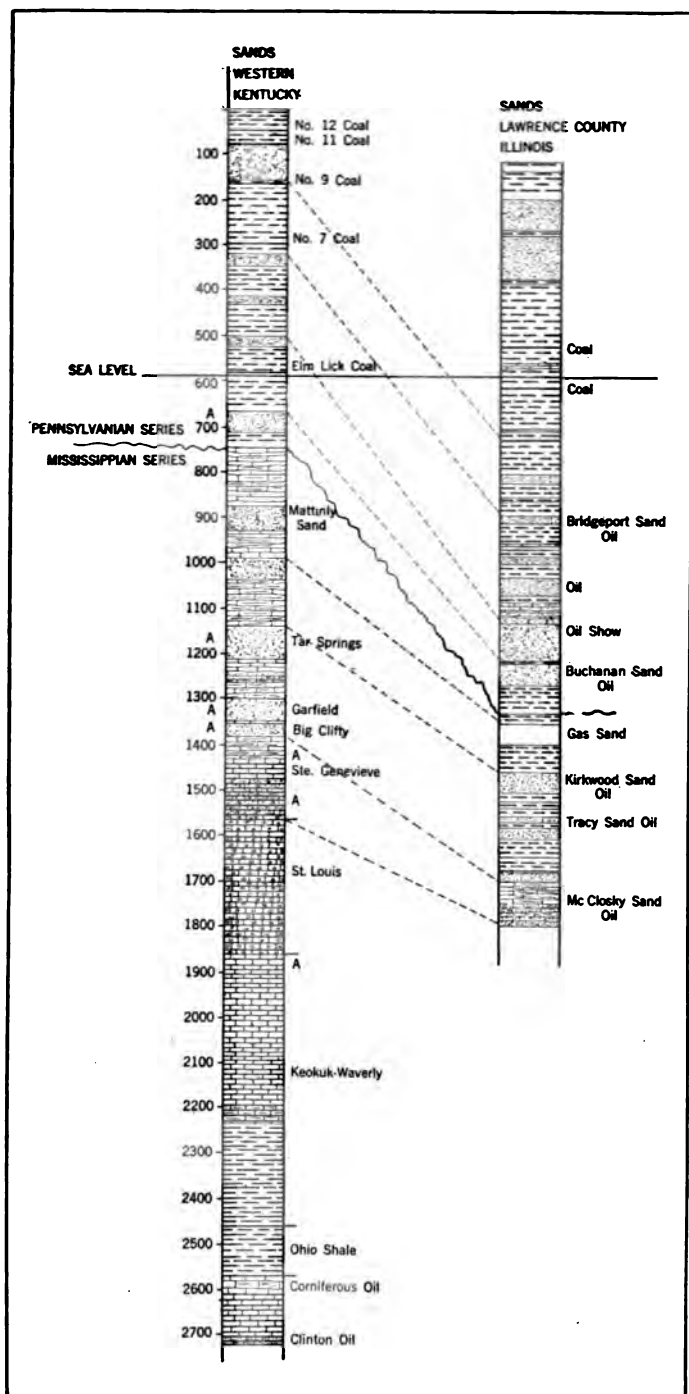


FIG. 1.—COMPARISON OF ILLINOIS AND KENTUCKY SANDS.

southern Magoffin counties. It is here that the best undeveloped areas in eastern Kentucky are to be expected.

Discussion of sands is here restricted to the five more promising districts:² (1) Breckinridge-Meads County; (2) interior western coal field; (3) southeastern border of western coal field, comprising western coal field and vicinity; (4) eastern flank of Cincinnati arch; (5) interior of eastern coal field, the last two comprising the eastern coal field and vicinity.

The western coal field and vicinity offer the following oil-bearing sands (see Fig. 1):

Pottsville Sands.—The Pottsville sands, represented by the Bridgeport and Buchanan sands in Illinois, give little promise in Kentucky owing to their shallow depth where the structure is favorable. The Pottsville sands, where outcropping in Edmonson County, are asphaltic.

Chester Sands.—The Chester group is represented in the Lawrence County (Illinois) field by two sands, but is here represented by five, two of which are invariably asphaltic on the outcrop, while the uppermost shows heavy salt-water saturation, which makes it promising for oil if tested under proper conditions. The Garfield and Tar Springs beds are asphaltic. The Big Clifty shows oil on structure. Where these sands are under ample cover, more than 500 ft., and of a thickness greater than 5 ft., on proper structure, they will probably prove oil bearing. These are quartz sands, and in the interior of the western coal field, at depths of 900 to 1,400 ft., should prove the best pay sands in Kentucky, but of small importance on the borders of the coal field.

Ste. Genevieve Oölites.—Where examined on the outcrop, notably on the south of this coal field near Bowling Green, these oölites carry volatile petroleum and are the equivalent of the McClosky sand of Illinois. This sand, at 1,400 to 1,600 ft. depth, should prove productive in the interior of the coal field.

Keokuk Sands.—Two productive sands are known, both dolomitic limestones, one near the top and the other 80 ft. above the Devonian or Ohio Black shale; neither of these sands has proved of commercial importance so far.

Corniferous, Niagara, and Clinton Limestones.—Each of these is 10 to 20 ft. thick and dolomitic. They offer good chances for oil and occur at depths of 1,600 to 2,000 ft., lying 30 to 200 ft. below the Ohio shale. The Carter sand of the Ohio County wells is in one of these limestones and is probably equivalent to the Boyds Creek sand (of Allen and Barren counties) of the Niagara limestone.

Cincinnati Rocks.—In the upper beds a sandy limestone suitable for an oil reservoir may occur, but it is doubtful.

²For details of logs and of sands see *The Oil and Gas Sands of Kentucky*, by J. B. Hoesing, *Bulletin No. 1, Kentucky Geological Survey* (1905).

Trenton Limestone.—In Indiana and Ohio this is one of the best productive horizons, containing 50 ft. or more of dolomitic limestone. In Kentucky, this limestone is calcitic with no dolomite beds and hence will not produce oil.

Oregon Bed.—Between the Trenton and Stones River groups, the Oregon bed, a dolomitic marble, outcrops in central Kentucky. There is nothing either for or against this bed proving productive, though on the whole we should advise against drilling deeper than the Clinton in either the eastern or western districts. The Calciferous sand would be too deep to consider drilling for in the western coal field or vicinity.

In the area west of the eastern coal field, among the oil-bearing sands are the following:

Keokuk Sand.—This is the gas sand of Wayne County, always dolomitic.

Waverly Sand.—This includes the Berea grit near the base of the Waverly. The Berea here is a fine-grained sandstone, 15 to 60 ft. thick; according to Mr. Gardner this is the Beaver sand of Wayne County.

Corniferous, Niagara, and Clinton Limestones.—These dolomitic limestones, chiefly the Corniferous, form the reservoirs in the Ragland, Camp-ton, and other pools, while the Clinton is probably the productive sand at Cannel City. These occur at depths of 350 to 1,700 ft. The Calciferous is to be sought only on the western edge of the eastern coal field, where oil has been found in the higher sands; it occurs at a depth of 1,500 ft. below the Clinton limestone.

In the central portion of the eastern coal basin these same sands occur, but at greater depths, and in addition the Big Injun sand occurs at the top of the Waverly, while the Pottsville or conglomerate group contains the Jones, Epperson, Beaver, and Salt sands or their equivalents, with only slight chances for oil at the base of the coal measures. The Keokuk sands either are too thin or are absent here. The Chester, well represented by sands in western Kentucky, contains here either only a thin and probably valueless sand, or none at all.

The dolomitic limestones of eastern Kentucky, though rarely more than 15 to 35 ft. thick, offer wells of small size but long life; it is from these that this section of Kentucky must get the bulk of its production, though the Calciferous will later doubtlessly yield oil and gas.

In Kentucky, as in other oil-producing States, the narrow oil belts parallel the more sharply folded and faulted areas, while as distance from the axis of the large uplift is gained the folds become more and more gentle, with possibilities for larger and broader oil fields. Thus the Barren-Allen County oil fields, representing only narrow streaks of oil, give place farther west to broad and more promising terraces, as yet untested, on the southeastern edge of the western coal field; within the latter, also, favorable areas exist of which only a few have been mapped.

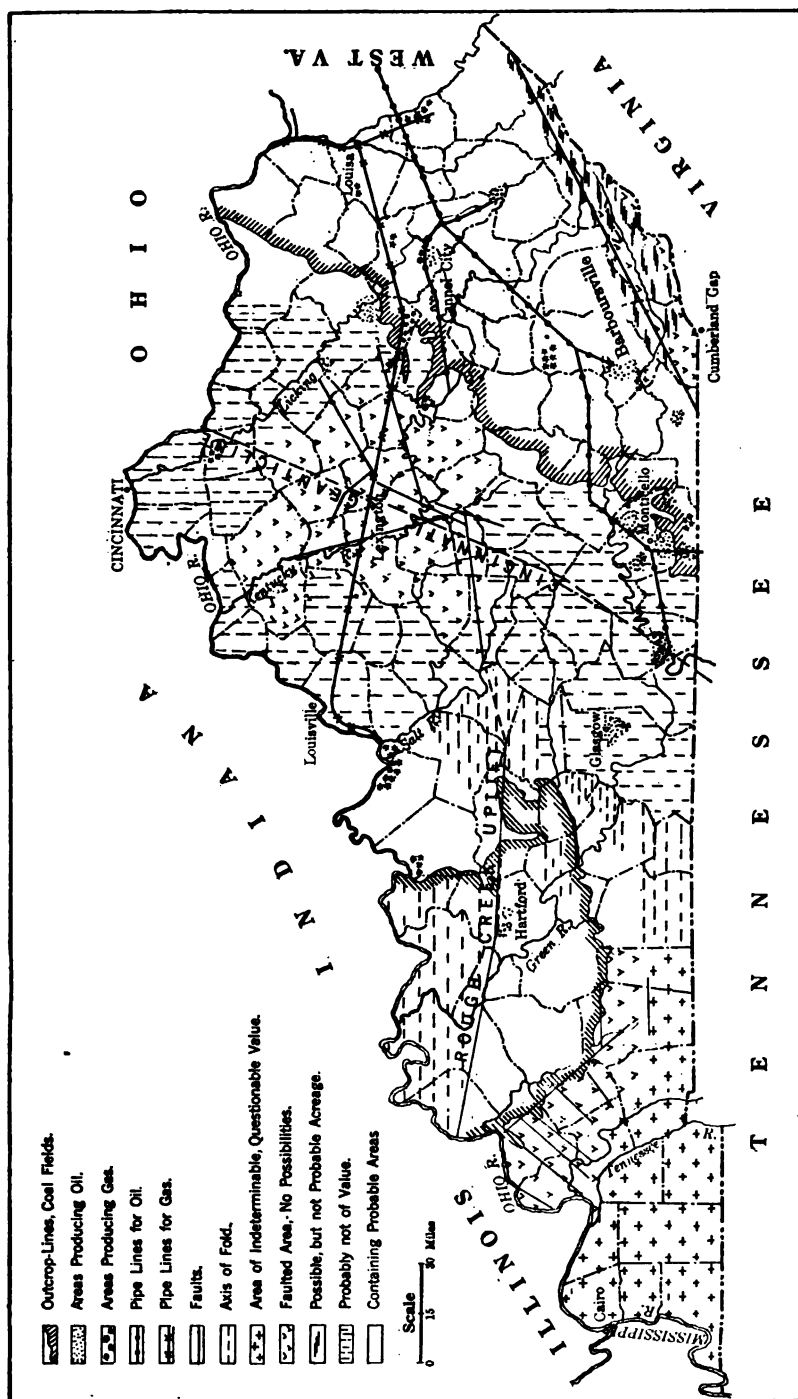


FIG. 2.—MAP SHOWING OIL AND GAS FIELDS OF KENTUCKY.

It will be seen from the accompanying map (Fig. 2) that one-third of the area of Kentucky, that portion chiefly within and adjacent to the coal fields, offers chances for oil pools. Some portions of this area are already condemned and it is unlikely in any event that after careful geological investigation more than one-hundredth part of this acreage will prove worthy of testing. In addition, other areas may show possibilities, while in some no adequate surface indications exist to show whether or not they are worthy of testing. It is needless to test the faulted areas, shown on the map, which have not ample shale to seal the faults.

Of the oil pools already developed, those of the Wayne County field have yielded the bulk of production; Ragland, Barbourville, Beaver Creek, Campton, Cannel City, Hartford, Diamond Springs, and Glasgow furnish smaller but steady yields. The oil, with the exception of the Ragland field, which because of its shallow depth is heavy and of low grade, is all of the green-oil type, some even being of the amber type, and all except the first having a gravity of 35° Bé. or higher. The largest wells have produced from the Corniferous and Silurian sands, the initial flow in a few wells at Hartford and Cannel City reaching 200 and 600 bbl. In the Wayne County pools, the Beaver sand occasionally reaches an initial output of 100 bbl. shot. Of the gas pools, the only ones producing commercially in eastern Kentucky are the Menifee, Martin, Floyd, Knox (Barboursville), and (Cannel City) Morgan County fields; while in western Kentucky small gas pools are developed in the West Point, Brandenburg, and Cloverport region of Meade, Breckinridge, and Hardin counties, near Central City in Muhlenberg County and at Diamond Springs in Logan County.

That Kentucky will upon proper development yield considerable oil there is little doubt. Since most of the pools are of small size, wild-cattng without careful geologic mapping in the possible areas would lead to heavy losses. Mr. Gardner and the writer mapped a few undeveloped favorable structures giving promise of pools of fair size. Owing to the prevalence of dolomitic sands of limited thickness, wells of more than 500 bbl. initial flow are apt to be rare, while most wells are likely to yield less than 100 bbl. More favorable results, if any, are, however, to be expected in the Chester sands of the interior of the western coal field. With most of its oil of high grade, Kentucky offers undeveloped fields of small daily output per well, but with good staying qualities, attractive to the oil producer rather than to the oil speculator.

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

A Modern Rotary Drill

BY HOWARD R. HUGHES, HOUSTON, TEXAS

(New York Meeting, February, 1915)

In drilling for water and oil to reasonable depths through the generally soft yielding clay and sand formation of the Coastal Plain of Texas, Louisiana, and Mississippi, the rotating method of drilling was adopted, principally on account of the easy and quick penetration, and the low cost of the drilling plant.

In favorable ground, free of heavy gravel and rock strata, as much as 1,000 ft. has been drilled in less than 36 hr., although such performances were of course rare.

Hydraulic rotary drilling, or, as it is now called, rotary drilling, was used in the above States as early as 1880. The plant consisted of an ordinary derrick, a 25-h.p. boiler, a small hoist, a steam pump, and a water swivel with hose attachment, and an ordinary flat diamond-pointed bit.

The successful drilling in of a phenomenal oil well by this process on Spindletop, near Beaumont, Texas, on Jan. 10, 1901, and the ascertained impracticability of drilling subsequent wells in the same locality by other methods (owing principally to heavy quicksand under pressure from below), brought rotary drilling into great prominence, practically to the exclusion of any other process throughout the Coastal Plain, and later on elsewhere.

The method, as its name implies, involves the rotation of a pipe by means of machinery placed on the derrick floor. A drill bit attached to the lower end cuts a clearance for the drill pipe, with much the same motion and effect as an auger. Water forced through the drill pipe by means of a pump, and escaping through the bit, removes the cuttings and returns to the surface outside the drill pipe. In this manner the hole is kept open, permitting the drill stem to rotate freely.

The pressure of the column of muddy water holds up the walls of the hole until it has been cased.

Practically all the wells of the Gulf coast region, numbering nearly 10,000, have been drilled with this system. During the last five years its use has been extended to many other States and countries.

The great drawback hitherto has been the slow progress made in drilling rock and other hard formations. For soft, caving formations no other system can approach it in efficiency.

The old style bit in general use is known as the fishtail type, as shown in Fig. 1. Having only two cutting edges it soon grinds down flat when hard rock is encountered. At times only a few inches a day can be made with it. The racking and wrenching to which the machinery and drill

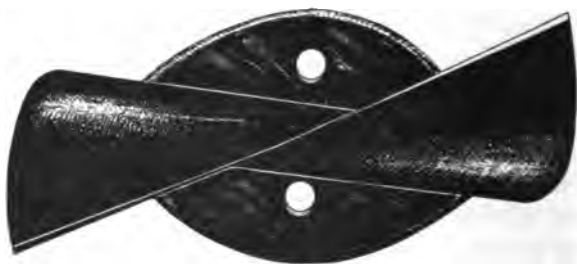


FIG. 1.—FISHTAIL BIT.

pipe are subjected, when drilling a hole from 4 to 18 in. in diameter, result often in twist-offs of the drill stem and costly fishing jobs. With the use of the heavier rigs and deeper drilling the need for a bit adapted to cutting rock became a crying necessity.

It was to meet this need that the cone bit, known as the Sharp & Hughes, was invented by me in 1908.



FIG. 2.—SHARP & HUGHES CONE BIT.

In brief, it consists of two or more detachable, cone-shaped cutters of hardened steel (see Fig. 2). These cutting cones revolve on bronzed bearings, lubricated with a special heavy viscous oil supplied by means of a small pipe carried inside the drill stem (see Fig. 3). The cutters, being detachable, may be removed and sharpened when dull.

The old style fishtail bit scraped its way through the rock encountered. With hard rock or sandstone it soon wore flat, lost all cutting power and had to be renewed. This necessitated the removal and replacement of the entire drill stem, with many (perhaps several thousand) feet of pipe, the work of many hours.

The principle of the cone bit is entirely different. The edges, or cone points of the bit, roll in a true circle like a cone bearing, and crumble or chip away the rock. The cone points, being of very hard steel, wear away slowly. Often they show but slight wear after drilling 50 ft. of rock, a few inches of which would completely dull the ordinary fishtail



FIG. 3.—BROADSIDE VIEW OF SHARP & HUGHES CONE BIT.

bit. The rolling motion allows the cutting edges on the cones to chip the rock, one edge after another.

Fig. 4 shows the bit, drill pipe, and lubricator in the hole, ready for drilling. The lubricator pipe, about 12 ft. long, is filled with oil, which is forced down into the bit by the pressure of the circulating water on the plunger. This figure shows also that the bottom of the drill hole as formed by the operation of the bit presents a perfect seat for a water-tight joint, preventing leakage after the casing has been set. When the cone bit is introduced in a hole to which previous use of the fishtail or diamond-pointed bit has given a V-shaped bottom, it must be advanced slowly

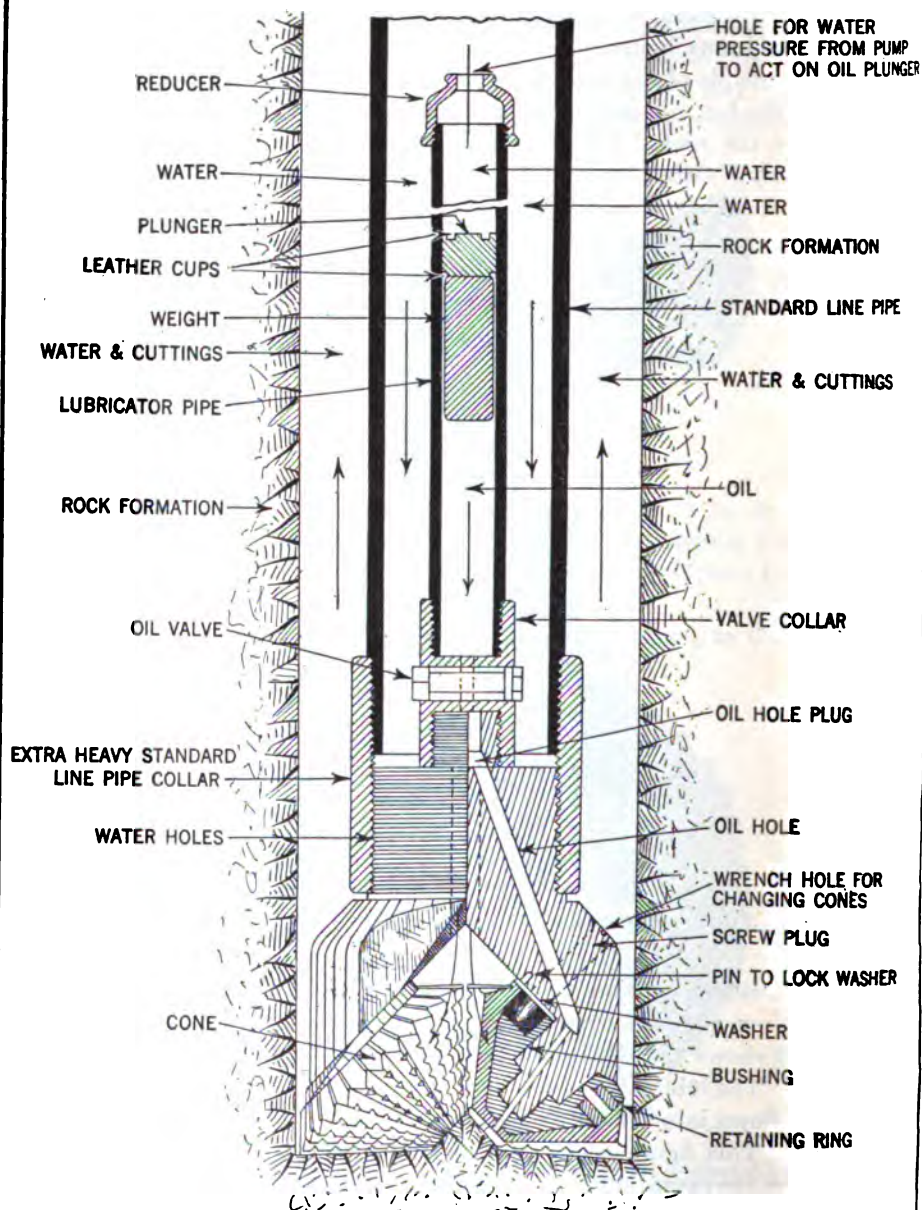


FIG. 4.—SHARP & HUGHES CONE BIT READY FOR OPERATION.

and carefully for the first foot, so that it may change that shape to suit its own form of cut.

Fig. 5 shows the whole drilling-plant and apparatus of this system.

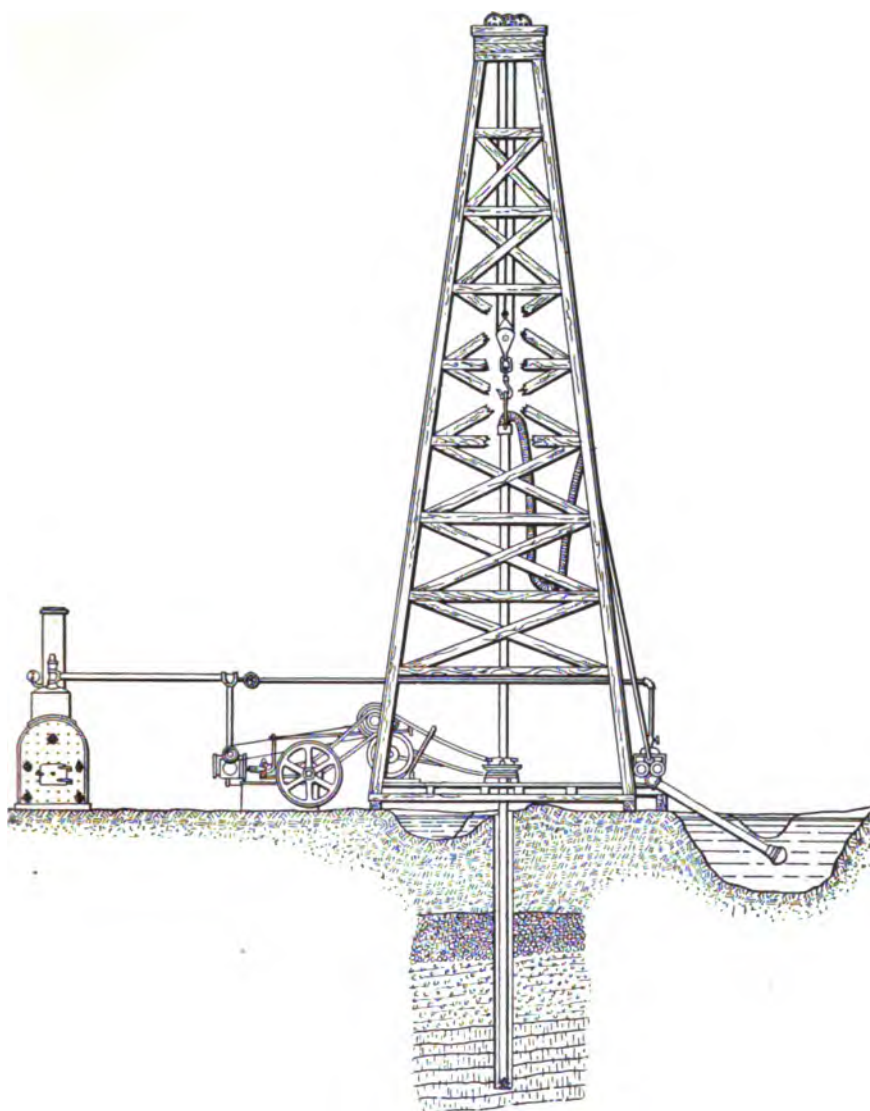


FIG. 5.—A COMPLETE ROTARY DRILLING OUTFIT.

The proper adjustment of the weight upon the bit is the secret of good work with this drill. Experience has shown that the following weights give satisfactory results for the corresponding bits of standard sizes:

Drilling Weight on Bit

Diameter of Bit, Inches	Weight, Pounds	Diameter of Bit, Inches	Weight, Pounds
2½	2,870	6½	7,900
2¾	3,480	7¼	8,650
3¼	4,700	7¾	9,270
4¼	5,310	7¾	9,560
4¾	5,910	8¼	10,300
5¼	6,670	9	11,000
5¾	7,140	9¾	12,000

For bits larger than those in the table, as much weight as practicable may be employed with little or no risk of overloading the bearings in the bit. But within the limits of the table, the weights given are probably as great as prudence would permit. For extremely hard rock, the speed, not the weight, should be increased.

The following actual working tests show the performance of the cone bit in comparison with the old fishtail:

The House No. 1 well of the Producers' Oil Co., Humble, Texas, struck rock at 1,819 ft., after which the fishtail bit bored 38 ft. in 19 days—an average of 2 ft. a day. The cone bit was then substituted, and bored 72 ft. in 6 days, or 12 ft. a day.

The comparative costs were as follows:

FISHTAIL*Day crew:*

4 Men, \$3 per day, 19 days.....	\$228.00
1 Driller, \$200 per month, 19 days.....	126.67

Night crew:

4 Men, \$3 per day, 19 days.....	228.00
1 Night driller, \$5 per day, 19 days.....	95.00
Dressing 38 9-in. fishtail bits at \$2.25.....	85.50
Steam and water, 19 days, at \$10.....	190.00

Total cost for 38 ft. of 9-in. hole..... \$953.17

Average cost per foot..... \$25.08

HUGHES CONE BIT*Day crew:*

4 Men, \$3 per day, 6 days.....	\$72.00
1 Driller, \$200 per month, 6 days.....	40.00

Night crew:

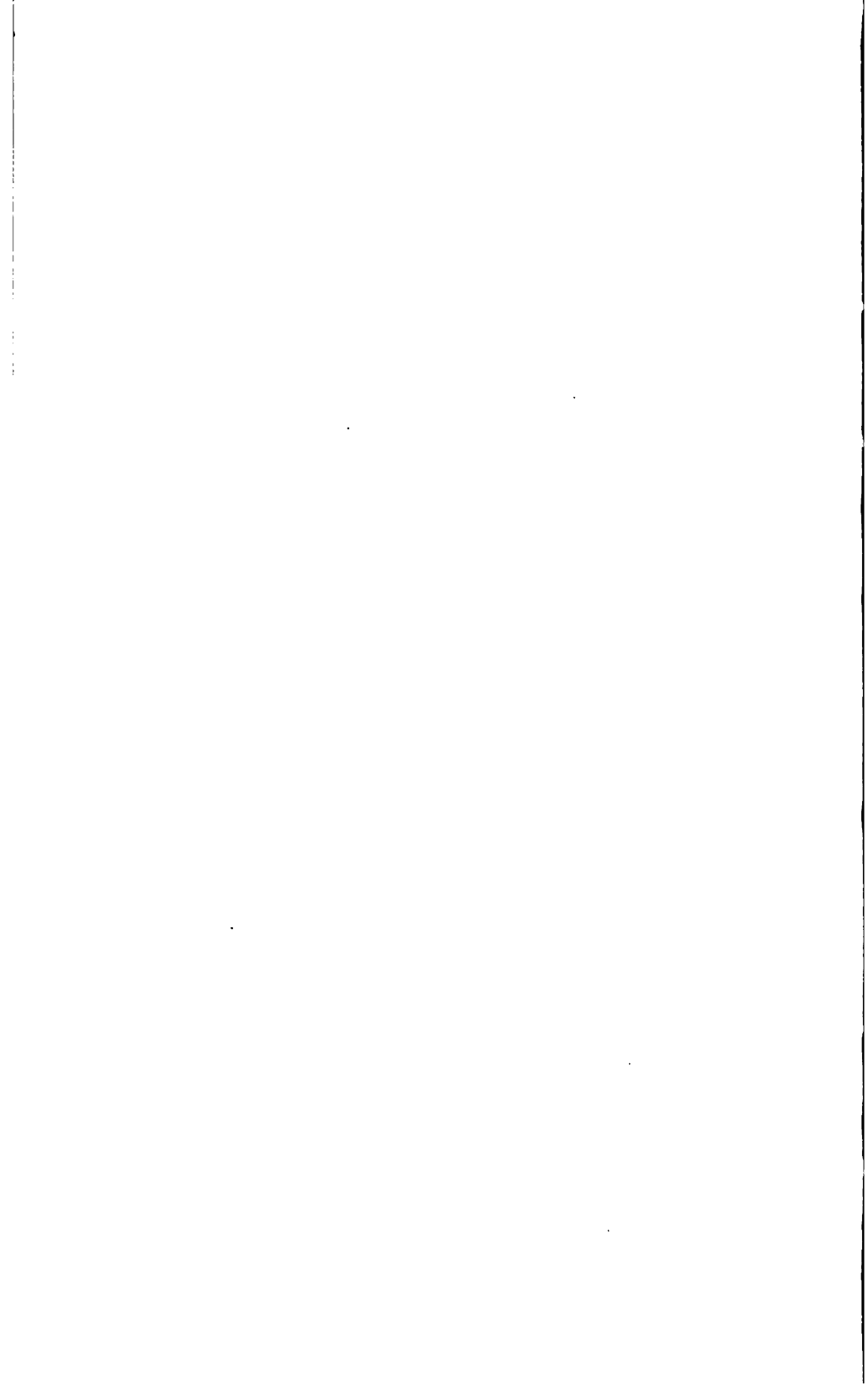
4 Men, \$3 per day, 6 days.....	72.00
1 Night driller, \$5 per day, 6 days.....	30.00
Steam and water, 6 days, at \$10.....	60.00
Rental of bit for 30 days.....	50.00
3 Sets of cones, at \$45.....	135.00

Total cost of 72 ft. of 9-in. hole..... \$459.00

Average cost per foot..... \$6.38

This comparison takes no account of the strain and wear upon machinery, the greater cost of superintendence per foot drilled, and the loss of time, under the old system.

While primarily designed for oil and water wells, this cone bit can be applied in drilling sump holes for mine pumps, in making air shafts, and in driving holes inclined at various angles from the vertical. Its use greatly enlarges the field of the rotary system, and the cone bit already is extensively used in California, Mexico, Trinidad, Roumania, Russia, Persia, Egypt, Japan, Borneo, and India, by presenting the great advantages of more rapid drilling through hard rock; of reaching greater depths than any other rotary apparatus can compass; of finishing a hole with smaller reduction of surface diameter than any other system permits; of the consequent requirement of fewer "strings" of casing; of less deterioration of drill pipe through strains and vibration, and of the saving of much time consumed in removing the drill pipe for sharpening or changing bits.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Dehydrating Oil Plant of Nevada Petroleum Co., California

BY S. J. HARDISON, COALINGA, CAL.

(New York Meeting, February, 1915)

IN the fall of 1912, the appearance of water in the oil of the Nevada Petroleum Co., Coalinga, Cal., made necessary the installation of a dehydrating plant to reduce the water below the 3 per cent. limit prescribed by the agency.

Unlike the mining industry, technical literature of the oil industry is limited and extremely unsatisfactory. Until the recent efforts of the Petroleum and Gas Committee of the American Institute of Mining Engineers, no concerted movement has been instituted to secure publication of papers dealing with the problems of the oil business, and because of the additional fact that dehydrating of oil has not been practiced in California to any extent up to comparatively recent time, it was found necessary to experiment in order to determine the most satisfactory plan for this purpose.

It is easy to write regarding successful enterprises, but, while not so pleasant, it is equally desirable to write of failures so that others may be saved the loss incident to such investments.

The water in this oil occurs both free and as an emulsion. Free water easily settles out, but the emulsion requires treatment. So far as can be determined, the emulsion consists of globules of water surrounded and enveloped by a film of oil. Starting with this hypothesis, the theory has been evolved that in order to break up this emulsion the water must be heated at least to boiling point, when an explosion takes place destroying the globule. Unfortunately, this condition was not recognized in the installation of the first plant, which was planned on the following lines:

A heater was arranged so that all the oil from the wells could be heated before reaching the shipping tanks, these being already fitted with coils for steam heating. Tanks were filled with about 4 ft. of water, live steam was turned into the coils and the water brought nearly to the boiling point, when the heated oil was run in. This for a time was partly successful, but the length of time necessary to apply this heat was too great for practical operating.

The arrangement of the heater plant was as shown in Figs. 1, 2, and 3. Three old boilers were suspended on substantial pipe supports, dry

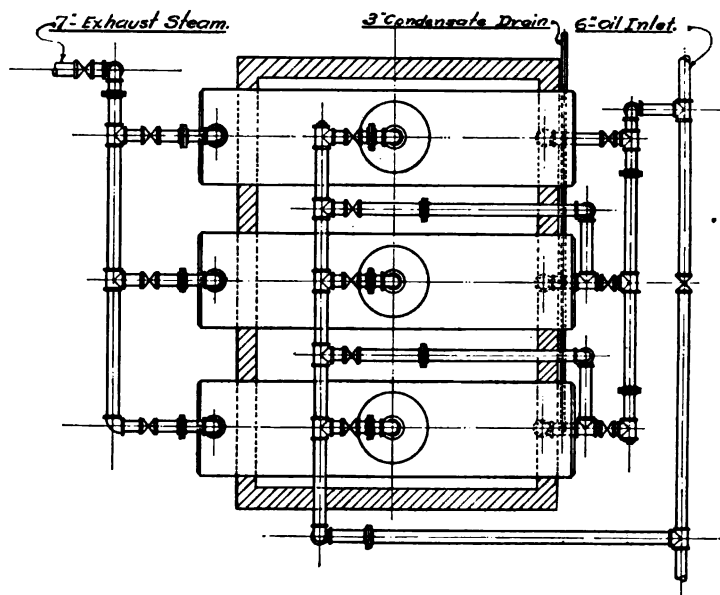


FIG. 1.—FIRST OIL-HEATING PLANT. PLAN.

brick walls filled up with oil sands being used to prevent radiation on bottom and side of boilers; the tops of the boilers were covered with bats and oil sand after the pipe work was done. Headers of galvanized

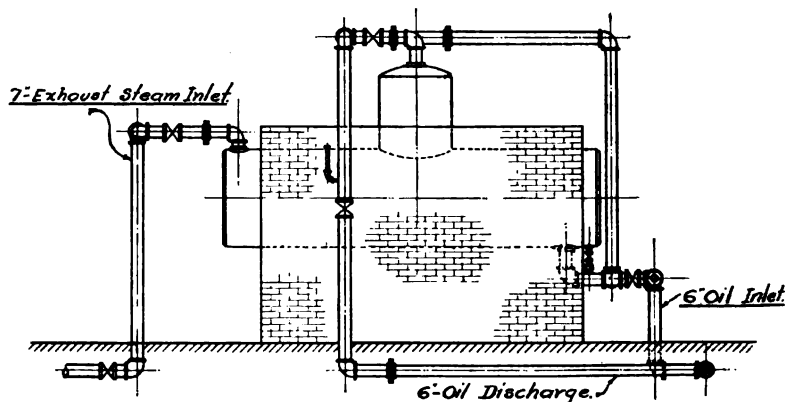


FIG. 2.—FIRST OIL-HEATING PLANT. SIDE ELEVATION.

iron were made to connect the inlet and the outlet steam. Exhaust steam from the compressor and pumps of the central plant was carried

through 7 $\frac{5}{8}$ -in. casing pipe, connected to the header and thus through the boiler tubes. The outlet was high enough for the condensed steam to gravitate back to the hot-water well of the central plant.

The plant was placed near the shipping tanks and close to the main line carrying oil from the wells to the shipping tanks.

The piping was arranged so that the oil could be run from the main through one, two, or all three boilers, usually traveling through all three; or the oil could be run direct through the line as before, in case of leakage in the plant. As no fire was used around this plant, there was no danger from that source.

This plant heated the oil to an average temperature of 170° F. very satisfactorily, and with practically no attention or cost, outside of installation. The loss of about 10° in transit to the tanks made about 60° to be supplied by the live-steam coils in the tanks. The water-heating

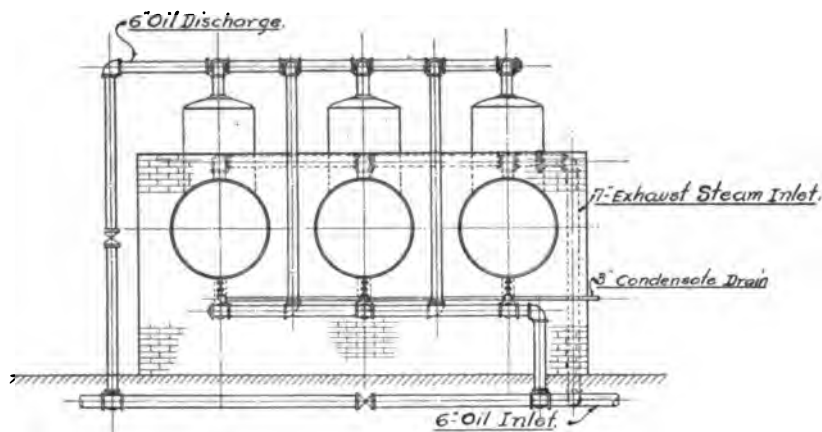


FIG. 3.—FIRST OIL-HEATING PLANT. END ELEVATION.

system was not adapted to the conditions and was discarded, the heat then being applied directly to the oil through the coils, but this was also too slow for the necessary operating capacity.

It is certain that the failure of this plant was due to insufficient heating of the oil, and a second plant was built with a view to correcting this difficulty. This plant was constructed as follows:

Two old 40-h.p. boilers were erected, with the typical oil-field mud setting, the boilers being suspended on pipe supports, with regular oil burners for supplying heat direct under boilers.

The piping was arranged so that oil could be run through either or both boilers, the oil being run in at the bottom and rear, and out at the dome; and again in at the bottom of the second boiler, through it and out to the storage reservoir, from whence it was pumped to the shipping



FIG. 4.—SECOND OIL-HEATING PLANT.

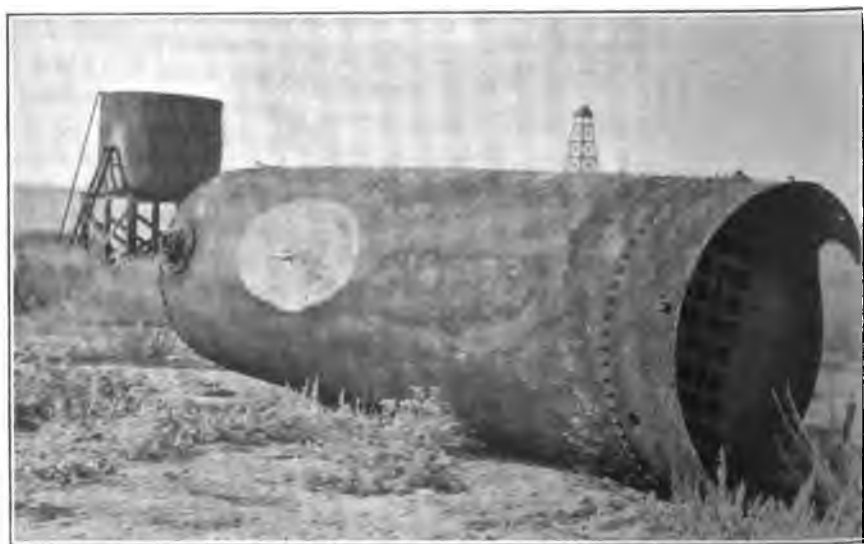


FIG. 5.—BURNT-OUT BOILER, SHOWING BAGGING.

tanks. The heater was still used for this plant, and when both boilers were being used a light fire was carried under the first boiler, the heavier fire being applied to the second boiler. The 6-in. overflow delivery

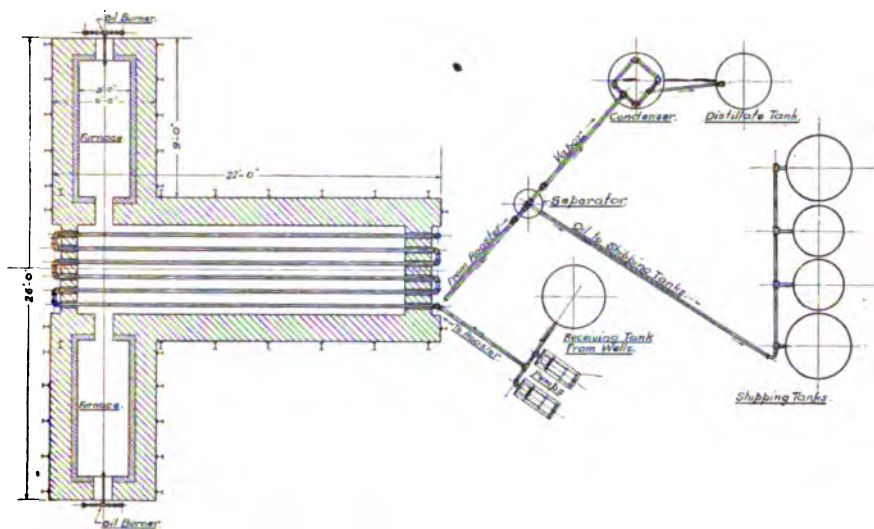


FIG. 6.—OIL-DEHYDRATING PLANT OF NEVADA PETROLEUM CO., COALINGA, CAL. PLAN.

pipe was set on a slight up grade to the receiving tank at the reservoir to insure a back pressure on the boiler.

The gauges usually showed a pressure of a few pounds and a tem-

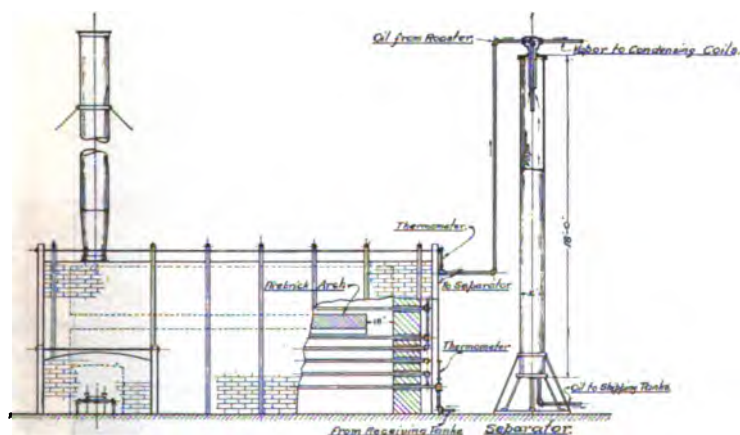


FIG. 7.—OIL-DEHYDRATING PLANT. SIDE ELEVATION.

perature from 230° to 250° , it being necessary to carry it above the boiling point.

All piping was 6-in., with no constricted turns or openings in the system. No attempt was made to recover any light products.

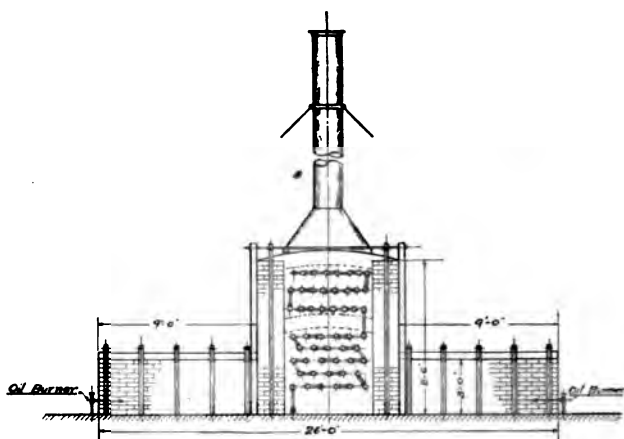


FIG. 8.—OIL-DEHYDRATING PLANT. END ELEVATION.

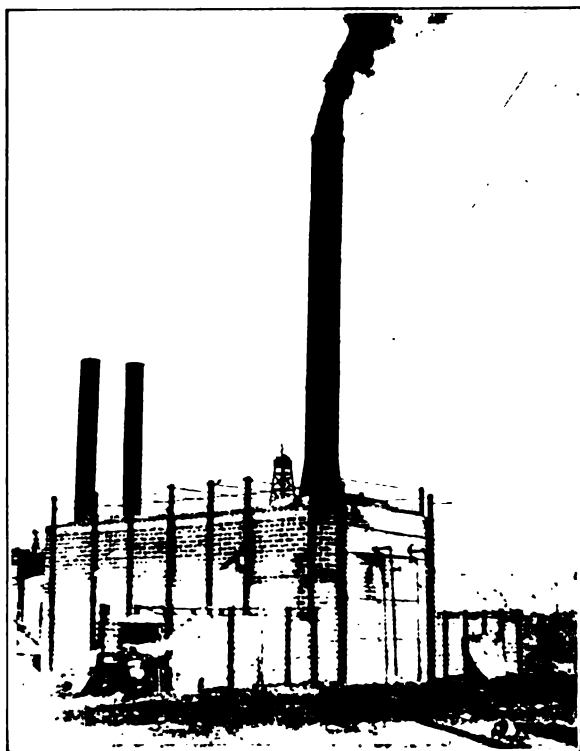


FIG. 9.—OIL-DEHYDRATING PLANT IN OPERATION.

While this plant did clean the oil, it was not a commercial success because of the excessive repairs, the sediment from the emulsion settling in the bottom of the first boiler causing blistering within a few days. These blisters had eventually to be cut out and patched and were the cause of several nasty fires owing to the boiler bagging and cracking and the oil running into the fire box. A burned-out boiler is shown in Fig. 5.

The experiment had now reached such a stage that certain definite conclusions were possible, viz:



FIG. 10.—SEPARATOR AND CONDENSING APPARATUS.

1. Exhaust-steam coils cannot be used successfully for heating the oil because the temperature is not raised sufficiently high.
2. If a separation of the oil and water is to take place, the temperature must be raised to a point that will generate steam and cause an explosion in the oil-enveloped globules of water, thereby liberating the water and permitting it to separate from the oil.

The third plant was then built, consisting of brick walls lined with fire brick inside of which were coils of 3-in. pipe through which the oil was pumped. These coils were constructed in horizontal rows of five

and six 20-ft. joints below the arch and nine joints in each of the three rows above, 54 joints in all.

Furnaces were built on either side of the front so that the flames would not come in direct contact with the coil. The heat from the furnace passes into the lower compartment, thence to the back and under the arch, then through the opening at the back, into the top compartment and out through the stack at the front end.

A receiver was erected for the oil as it left the roaster, the function of which is to separate the vapor from the oil and to recover the light products that would otherwise pass off into the atmosphere and be wasted. The oil passes out at the bottom of this receiver and into the shipping tanks, the vapor passes out of the top through a coil submerged in cold water, where it is condensed and from which it is allowed to collect in a receiving tank.

This last plant, illustrated in Figs. 6 to 10, has proved a complete success. Because the coils are small, the solids from the oil cannot precipitate and form blisters. It has been operated daily for the past year without any repairs.

The cost of operating the dehydrating plant per 100 bbl. treated is as follows: Labor, 75c.; fuel consumed, 74c.; steam consumed, 22c.; total, \$1.71. The condensation per 100 bbl. is: Water, 10.40; light oil, 0.43; total, 10.83. The gravity of the light oil is 37.2. The oil before treating showed: Temperature, 80° F.; specific gravity, 14.8; B. S. and M., 11.6 per cent. The oil after treatment analyzed: Temperature, 260° F., specific gravity, 15.5; B. S. and M., 1.0 per cent.

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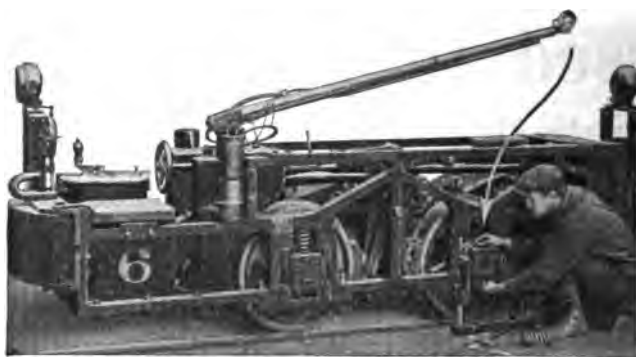
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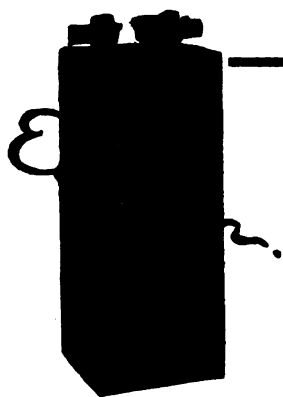
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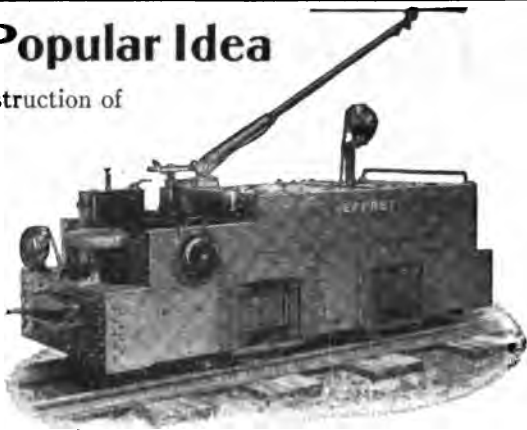
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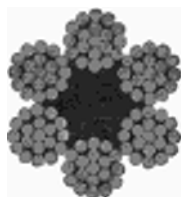
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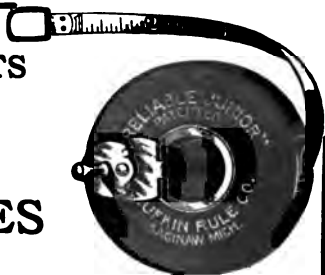
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[illegible]

Signature.

ARTICLE II.—MEMBERS.

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

No. 100

APRIL

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Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

DUES PAYABLE

Attention is called to the following excerpt from the Constitution, Article III, Section 1:

"If any Member, Junior Member or Associate is in arrears for four months, the *Bulletin* shall no longer be sent to him and he shall be notified by the Secretary of the Institute that no publications will be sent to him until his arrears be paid."

In accordance with this provision the May *Bulletin* and following numbers will be sent only to those who have paid their dues for 1915. Those who have not sent in their 1915 dues are earnestly requested to do so, so that their names may not be taken from the mailing list, and that there may be no interruption in the sending of the Institute's publications.

SAN FRANCISCO MEETING

The 111th meeting of the Institute for the presentation and discussion of technical papers will be held in San Francisco, Cal., Sept. 16 to 18, 1915. A special train for members of this Institute and for the members of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, the Society of Naval Architects and Marine Engineers, and the International Engineering Congress, will be run from New York City to San Francisco, leaving New York on Sept. 9.

All papers to be presented at this meeting of the Institute must be in the hands of the Secretary on or before June 1, 1915.

BOARD OF DIRECTORS

Meeting of Feb. 16, 1915.—The Committee on Membership was appointed with the following personnel: John H. Janeway, *Chairman*; Karl Eilers, Lewis W. Francis, Louis D. Huntton, and Thomas H. Leggett.

President Saunders announced the appointment of the Committee on Increase of Membership, as shown on p. xxviii of this *Bulletin*.

The reproduction by the anastatic process of 100 copies of Volume IX of the Institute *Transactions* was authorized.

The sum of \$50 and the exchange of the *Bulletin* was awarded to the commission publishing the *Annual Table of Constants*.

The sum of \$49.07 was contributed to the expense of the John Fritz Medal Board of Award for the year 1914, and the Institute agreed in the future to share the annual expense for the John Fritz Medal Board of Award not to exceed one-quarter of \$500 in the year.

Upon the recommendation of the Committee on Membership, 36 Members, 1 Associate, and 13 Juniors were elected.

The resignations of 26 members were accepted.

L. D. Ricketts was elected a Vice-President of the Institute to serve the unexpired term of W. L. Saunders.

George D. Barron was elected a Director to serve the unexpired term of L. D. Ricketts.

Philip N. Moore was elected First Vice-President.

George C. Stone was elected Treasurer.

Rossiter W. Raymond was elected Secretary Emeritus.

Bradley Stoughton was elected Secretary.

Burr A. Robinson was appointed Assistant Secretary.

The following Executive Committee was appointed: William L. Saunders, *Chairman*; Benjamin B. Thayer, Sidney J. Jennings, Joseph W. Richards, and George D. Barron.

The Committee on Papers and Publications was appointed, the personnel of which is given on p. xxvii of this *Bulletin*.

The following Finance Committee was appointed: Charles F. Rand, *Chairman*, Albert R. Ledoux, and George D. Barron.

The President appointed the following Library Committee: E. Gybbon Spilsbury, *Chairman*; Karl Eilers, John Hays Hammond, Alex. C. Humphreys, and Bradley Stoughton.

James F. Kemp was unanimously elected an Honorary Member of the Institute.

MOTION PICTURE FILMS AND LANTERN SLIDES OF THE U. S. BUREAU OF MINES

Attention is called to the list of films and slides maintained by the U. S. Bureau of Mines for loaning purposes in connection with lectures and talks on mining and metallurgy. The Institute has separate copies of this list which it will be glad to send to persons who apply for them and will also be glad to receive information in regard to films, which information will be disseminated where it will be of use to those engaged in the industry and those who are preparing themselves to be engaged therein.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period Feb. 10 to Mar. 10, 1915:

W. R. Van Liew, Batoum, Russia.
Olof Wenstrom, Boston, Mass.
D. G. Miller, Denver, Colo.
E. E. Olcott, New York, N. Y.
Francis R. Pyne, Elizabeth, N. J.
Charles W. Goodale, Butte, Mont.

Edward S. Jones, Scranton, Pa.
Scott Turner, Tromso, Norway.
A. H. Boyd, Denver, Colo.
M. M. Thompson, Morsemere, N. J.
F. C. Merry, Brooklyn, N. Y.

Charles A. Filteau has been appointed General Manager for the Inca Mining & Development Co., Province of Carabaya, Department of Puno, Peru.

Charles T. Kruse has been appointed Superintendent of the Jones & Loughlin mines in the Marquette and Menominee ranges.

Harry Huntington Miller has been appointed manager for the Cia. A. Minera Lo Increible, at its mines in El Callao district, Venezuela.

Fred G. Farish has been appointed manager of the Lluvia de Oro Gold Mining Co.'s mines in Chihuahua, Mexico.

Percy E. Barbour has joined the editorial staff of the *Engineering and Mining Journal*.

Henry Hay has accepted a position at Chuquicamata, Chile.

Lee Olds Kellogg, for several years on the editorial staff of the *Engineering and Mining Journal*, is now Associate Editor of the *Engineering Magazine*, 140 Nassau St., New York.

W. C. Green, well known to the Western mining trade for the past 25 years, and for the past six years representative of the mechanical goods department of the Diamond Rubber Co., died Feb. 13, from an attack of pneumonia. Mr. Green's work for the Diamond Rubber Co. will be carried on by C. A. Tracy.

Homer L. Carr has returned to New York after six months spent in Honduras examining placer ground.

Adam R. Gordon has held the position of Manager of the New York & Honduras Rosario Mining Co., San Juancito, Honduras, since Sept. 1, 1913, before which time he was Superintendent of Mines.

Walter M. Dake, Jr., has resigned as Mill Superintendent of the Rainbow Mining Co., Rye Valley, Ore., and has been succeeded by L. A. Walker.

Dr. James Douglas was made an Honorary Member of the American Mining Congress at a recent meeting of its Board of Directors. The diploma was presented to him at a luncheon at the Engineers' Club on Feb. 17, 1915.

Dr. John C. Branner has submitted his resignation as President of Stanford University, to take effect July 31, 1915.

Dr. Willet G. Miller, Provincial Geologist of Toronto, has been awarded the gold medal of the Institution of Mining and Metallurgy of London.

Ricketts & Banks announce the dissolution of the copartnership heretofore existing between **Dr. Pierre DeP. Ricketts** and **Dr. John H. Banks**. Each will carry on the same line of business independently—the address of **Dr. Ricketts** being 80 Maiden Lane and that of **Dr. Banks**, 61 Broadway.

Philip S. Morse has resigned as Manager of the mines, mills and smelting works of the Sulphide Corporation in New South Wales.

Robert H. Richards was presented with the gold medal of the Mining and Metallurgical Society of America in recognition of his services in the department of ore testing at a dinner at the Chemists' Club on Thursday, Mar. 18, 1915.

Edward T. Stotesbury has been elected Chairman of the Reading Iron Co., Reading, Pa., to succeed the late **George F. Baer**.

Alfred C. Nelson, formerly with **A. G. McKee**, has opened offices at 418 Rockefeller Building, Cleveland, Ohio, as a consulting and contracting engineer.

T. A. Bennett, for five years in charge of the conveyor and elevator belt sales of the **B. F. Goodrich Co.**, Akron, Ohio, has resigned to become assistant to the General Sales Manager of the **New Jersey Zinc Co.**, with offices in New York City. **James D. Mooney**, formerly special mining representative of the Goodrich company, will succeed **Mr. Bennett**.

POSITIONS VACANT

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons.)

General manager for a pyrite mine in Virginia, with experience in the mining and particularly in the washing of pyrite, required. Salary at start, \$250 per month with perquisites. No. 28.

Assayer and cyanide man for a Costa Rica plant. Salary \$100. No. 29.

Mine manager to develop gold-mining property in Witwatersrand, Transvaal. Salary £2,000 to £4,000, according to the experience and ability of the engineer. No. 31.

Mine superintendent for a mine in Arizona containing copper, gold, silver ore. Will have entire charge of operation and development. Salary to start, \$200 per month with perquisites. No. 32.

Mining engineer to make complete report and suggest necessary machinery for development and operation of a placer property; also to examine a copper property. Both properties in Peru, South America. No. 34.

Research fellowships, carrying an annual stipend of \$500, open to students of approved universities and technical schools, are maintained by the University of Illinois. Appointments are made for two consecutive collegiate years and lead to Master's degree. For additional information address, The Director, Engineering Experiment Station, University of Illinois, Urbana, Ill. No. 35.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate in mechanical and mining engineering (1907), experience in Colorado, Montana, Mexico, Central America, in surveying, geological, assaying, chemical, and electro-metallurgical work; making concentration, flotation, and cyanide tests on ores; as superintendent; examining and reporting on properties. Speaks Spanish, French, and German. No. 196.

Member, technical graduate, aged 35 years, unmarried, 11 years' experience covering mine surveying, geology, assaying, sampling (including churn-drill sampling), and copper concentration, including two years' experience in mine examination work as field engineer in the West. Is thoroughly competent to report on any base-metal mine. Would accept temporary examination work or permanent position. No. 197.

Young lady with training in assaying, analytical, industrial, organic, physical, and metallurgical chemistry, desires a position as laboratory assistant. Will go anywhere a woman can go. No. 198.

Member, aged 29, married, technical graduate, five years' varied experience from mucker to assistant superintendent of mine and mill, experienced assayer, surveyor, and ore tester, also experience in hydro-metallurgical processes, open for engagement. No. 199.

Junior member. Technical graduate. Has worked up from the shovel to present position of efficiency engineer. Would like a position in the operation end where there is a good opportunity. Willing to go anywhere. No. 20.

Member, technical graduate, aged 34; 14 years' experience as assayer, surveyor, mining engineer, superintendent, and manager of gold mines. No. 203.

Member, technical graduate, married, practical experience prior and subsequent to graduation, in cyanidation, metallurgical research, surveying, mining, and milling, desires permanent position where ability to apply practical and technical principles can be appreciated. No. 204.

SOUTHERN CALIFORNIA LOCAL SECTION*Executive Committee*

THEODORE B. COMSTOCK, *Chairman*

SEELEY W. MUDD, *Vice-Chairman*

FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg.,
Los Angeles, Cal.

A. B. CARPENTER

C. COLCOCK JONES

A meeting of the Southern California Local Section of the Institute was held on Feb. 24, at the Hotel Clark, 21 members being present. In the absence of the Chairman and the Vice-Chairman, C. Colcock Jones presided. W. L. Austin of Riverside gave a very exhaustive and interesting talk on The Hydrometallurgy of Copper, which was followed by an informal discussion of the subject.

F. J. H. MERRILL, *Secretary*.

JAMES FURMAN KEMP, HONORARY MEMBER

At the meeting of the Board of Directors of this Institute on Feb. 16, 1915, the 14 members of the Board present unanimously elected Professor Kemp an Honorary Member. The nomination of Professor Kemp was signed by 21 members of the Institute, none of whom are members of the Board of Directors, and recited his qualifications for Honorary Membership in the following terms:

Professor Kemp graduated from Amherst College in the class of 1881, and from the School of Mines, Columbia University, in the class of 1884. After serving as instructor in geology in Cornell University for a few years he was called back to Columbia as adjunct professor in 1891, was made full professor the following year, and has since filled that office with dignity and honor. Among the first to recognize the importance of igneous rocks in the formation of orebodies, his studies have contributed greatly to that increase of knowledge of economic geology in which the *Transactions* of the Institute has had an influential part, and he is recognized both abroad and in this country as one of the most distinguished scholars in geology which America has produced. This recognition is evident in the honorary degrees of Sc. D. conferred on him by Amherst College in 1906, and LL. D. conferred by McGill University in 1913, as well as by election to membership in the National Academy of Sciences, and a corresponding member of the Academy of Sciences of Christiania, and the Geological Society of Stockholm. Professor Kemp was a Manager of the American Institute of Mining Engineers from 1896 to 1898, Vice-President from 1903 to 1904, a Director from 1905 to 1912, President and President of the Board of Directors in 1912, and Past President since that time. During these years he has labored loyally for the upbuilding of the Institute, serving on many committees, as well as in the offices enumerated. He has also served as President of the American Association for the Advancement of Science, the New York Academy of Sciences, and on the governing board of the Geological Society of America and the New York Botanical Gardens. His service is not confined, however, to these labors for the good of the profession and his contributions to exact knowledge of geology. In his almost 25 years as a professor in Columbia University there have come under his influence a great group of young men in whom he has inspired unflagging zeal in the search for truth, and to whom he has set an example of unselfish service. The value of such a service cannot be overestimated and in electing him to Honorary Membership the Institute would officially recognize the measure of appreciation which his professional colleagues have long accorded to him.

AFFILIATED STUDENT SOCIETY NOTE

At a meeting of the Colorado School of Mines Scientific Society held Feb. 26, 1915, the following officers were elected: V. D. Howbert, President; C. E. Carstens, Vice-President; F. E. Briber, Corresponding Secretary.

REPORT OF COMMITTEE ON INCREASE OF MEMBERSHIP

During the year this Committee has had the benefit of the services of 75 Committee members, resident in the various States of the Union and in foreign countries, 10 of them representing us abroad. A number of plans for awakening the interest of possible new members have been carried out and the results have been very gratifying. The comparative results for 1913 and 1914 are as shown in the following table:

	1913	1914
Applied for membership	630	946
Elected	450	1,006
Accepted election	433	921

When it is remembered that the depression following the outbreak of war in Europe has had more serious effect on the mining industry than almost any other, these results cannot be regarded as anything other than very encouraging. Because of the unsatisfactory conditions still prevailing it will doubtless be difficult to make even as good a showing next year. It is considered, however, that the function of this Committee is not merely to secure new members, having no regard for other than numbers; its true function is to exercise a proper supervision over the growth of the Institute, building it up in those fields in which it is weak, and directing its growth in such a way as to make it most effective in fulfilling its proper function as the national organization representative of the mining profession.

The plan of inviting the alumni of the leading mining schools to apply for membership, which was initiated last year, was completed this year, with the exception of one or two schools, notably the University of California, from which we have not been able to obtain a list of alumni. With these exceptions we have now extended to the graduates of all the important mining schools of this country, and several of those abroad, an invitation to apply for membership in the Institute. It is not thought desirable to repeat this invitation too frequently, and it is therefore suggested that this field be left alone for a year or two, when, if the mining industry experiences a marked revival of interest, it will be well to send out a second letter to the same men.

The activities incident to the meetings of the Institute, especially those outside of New York, are a fruitful field for awakening interest of possible members, and both at the Salt Lake meeting and the Pittsburgh meeting the result of the sessions was to encourage the sending in of applications by non-members who had been invited to attend the sessions. This work is the most visible in its effect, but probably the most important service rendered is by the earnest members of the Committee who throughout the year do not fail to take the opportunity to bring the Institute and its services to the profession to the favorable attention of the non-members in their respective States. For this reason I have suggested that it will be well only to ask the members to serve on this Committee for one year, replacing them with an entirely new Committee each year. In this way the work will have the effort of the differing circle of acquaintances of the new Committee members, and furthermore, it is somewhat embarrassing for a Committee member to be perpetually regarded as one who is endeavoring to secure applications. A man who only has to serve for one year will ask freely for applications during that year, but a man

who had to serve continually on this Committee might reasonably be expected to become somewhat sensitive on the subject of continually approaching his friends in this manner.

At the request of the Secretary of the Institute I have undertaken to organize the Committee for 1915. During the carrying out of this plan it proved that there was a third advantage which had not been foreseen, in placing new members on the committee. The letters received from those who have been invited to serve in most instances indicate their pleasure at being asked to engage in some useful work on behalf of the Institute, and since it is certain that we can count on the continued interest and loyal support of the old Committee members, the organization of a new Committee has therefore served to strengthen the interest in the Institute's work and to bring to attention the services of a fresh group of men who doubtless were willing to serve but had no opportunity for doing so. It will be noticed from the list, published elsewhere in this *Bulletin*, that many of the men who have been asked to serve are the younger engineers. These men were chosen because it is natural to expect that the greater number of new members will come from among the younger engineers of the profession, and also these men are more likely to have leisure to devote to the activities than men of more mature age whose time is more fully taken up with important business interests.

In the work of the Committee, matters have been so organized that as little labor as possible falls on the individual Committee members, and as much as possible of the routine of letter writing is performed by the Institute offices. This has proved very satisfactory and doubtless further progress in this direction can be made during the coming year. A new method was tried during the summer, by sending out to some 5,000 men engaged in the petroleum industry a well-designed and carefully printed folder drawing attention to the work of the Institute in this field. This was followed up in some instances by personal letters, in the writing of which Dr. David T. Day and Capt. A. F. Lucas kindly assisted the Committee. I regret to say that so far as can yet be seen the result of this work was rather disappointing and the cost of it was rather heavy. For the time being, therefore, it does not seem desirable to continue this or to extend the method to cover other fields of the mining profession. It is possible that further experience will show that these circulars have been efficacious in awakening interest in the Institute's work in a field where its activities are only just becoming known, even though they did not bring in any immediate applications. It is expected during the coming year to utilize to the full opportunities which will present themselves at the Engineering Congress in San Francisco and at the same time to awaken more interest in the Institute's work in California, where, as yet, comparatively little activity has been evident for reasons which are beyond the Committee's control.

THOMAS T. READ, *Secretary*.

INTERNATIONAL ENGINEERING CONGRESS

Members of the Institute seem to be under a misapprehension as to the relation between the Institute and the International Engineering Congress to be held at San Francisco, Sept. 20 to 25, inclusive, this year.

The Institute is one of the five national societies that joined in organizing the Congress, the others being the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, and the Society of Naval Architects and Marine Engineers. Each society has pledged from its general funds or some special fund a certain sum to guarantee the success of the Congress. To this the engineers resident on the Pacific Coast added \$13,000, the total guarantee fund amounting to \$37,000. It is hoped that this money, in part or in whole, can be repaid to the guarantors of the Congress. The sums necessary to conduct the latter and to publish the proceedings, are to be derived from membership fees paid equally by all participating. A member of the Institute as such is not a member of the Congress and will not receive any of the publications of the latter. He is eligible to become a member of the Congress and it is hoped that he will do so. The membership fee is \$5, and this entitles one to receive the introductory volume of the Congress containing the proceedings, a digest of the other volumes and a general index. It also entitles the member to one other volume of the transactions, to be chosen as he will. For moderate sums he can buy additional volumes, the whole set costing delivered, \$30 or less.

On Jan. 12 the total membership was 2,350, of which only 290 were recorded as mining engineers. Full details regarding the Congress have been sent to every member of the Institute, and the failure of the members to take a more active interest in the matter is believed to be due to some misunderstanding of the situation.

The Institute is represented on the Board of Management by its President, Secretary, and four members, resident in San Francisco. Mr. Saunders, President of the Institute for 1915, is at the same time Chairman of the Inter Society Reception Committee in New York. The Institute itself will have a large part in the program, not only in the sections devoted specifically to mining and metallurgy, but also its members will contribute to the volumes on materials of construction and other topics. Among those preparing papers are W. H. Shookley, W. W. Mein, R. V. Morris, E. T. Lombardi, Sidney J. Jennings, A. Chester Beatty, D. C. Jackling, W. V. Winchell, J. A. Holmes, and J. Parke Channing.

The metallurgical sessions will be devoted to a review of the present status and prospects of the art as related to the different metals. Among those assembling material for this part of the program are H. M. Howe, E. P. Mathewson, W. R. Ingalls, E. F. Roeber, C. H. Fulton, H. O. Hofman, William Campbell, R. H. Richards, and others. Every effort is being made to secure comprehensive papers and adequate discussions, and the material already in hand assures a notable program.

Members of the Institute who have not yet sent in their application for membership in the Congress are urged to do so at once, so as to enable the committee in charge to plan intelligently for both the sessions and the work of publication. Write to W. A. Cattell, Secretary, Foxcroft Building, San Francisco, Cal.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining and Metallurgy

MINING COSTS OF THE WORLD. A compilation of cost and other important data on the world's principal mines. By E. N. Skinner and H. R. Plate. New York, McGraw-Hill Book Co., 1915. Price \$5. (Gift of Publisher.)

[NOTE.—This volume of 406 pages, with 9 maps, contains a compilation of the costs of mining and the local conditions influencing such costs at about 325 metal mines, scattered throughout the world. Since these are the principal producers of the world's metal supply, and therefore represent operations on a large scale, backed by adequate capital, conducted with technical skill, and with more or less detail in recorded periodical official reports, the figures here collected constitute a standard by which to measure the cost of mining. Direct comparisons would, of course, be unfair, if no account were taken of the influence of local conditions, such as the nature and size of the deposit, method of exploitation, rate of wages, etc. But many of these elements of

the problem are supplied by the compilers under the table of costs for each mine, so that it is often possible to detect at once the conditions which warrant a high cost, otherwise liable to be ascribed to lack of economy in management. One thing, of course, is plain. The costs at small or undeveloped mines are certain to be greater per ton than in such operations as are here described; and this evident consideration furnishes for the benefit of both engineers and investors a check upon the estimates of promoters. There are many other ways in which this book will be useful, aside from the fact that, as a condensed survey of a world-wide industry, it will be intensely interesting to experts. Its tables cover, as a general rule, the operations of 1913, and the authors deserve much credit for the industry and care with which they have executed so great a labor, the tact with which they have secured the aid of other experts, and the frankness with which they indicate the shortcomings to be remedied in future editions.—R. W. R.]

THE CYANIDE HANDBOOK. Ed. 2. By J. E. Clennell. New York, McGraw-Hill Book Co., 1915. Price \$5. (Gift of Publisher.)

[NOTE.—This second edition has brought the information up to date. There are many new sections and new illustrations. Particular attention is given to the chemical aspects of the question. The book is thoroughly indexed.—W. P. C.]

BETRACHTUNGEN ÜBER DIE WÄRMEBILANZ EINES SIEMENS-ZINKOFENS. By Fr. Eulenstein. Halle, n. d.

KARBIDE UND SILIZIDE. By Otto Hönigschmid. (Monographien über angewandte Elektrochemie. Band XLV.) Halle a. S., 1914.

ÜBER BOHRER FÜR GESTEINSBOHRMASCHINEN. By Arthur Gerke. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 145.) Kattowitz, n. d.

INTERMETALLIC COMPOUNDS. By C. H. Desch. New York, 1914.

DIE RÖSTUNG DER ZINKBLENDE UND DIE THERMOCHEMISCHEN VORGÄNGE BEI DIESEM METALLURGISCHEN PROZESS. By A. Rzehulka. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 146.) Kattowitz, n. d.

MANUEL PRATIQUE DE FONDERIE. By J. Duponchelle. Paris, 1914.

Geology and Mineralogy

BEITRÄGE ZUR KRYSTALLOGRAPHIE UND MINERALOGIE. Band 1, pts. 1-2. By V. Goldschmidt. Heidelberg, 1914.

GEOLOGICAL AND STATISTICAL MAP OF BRAZIL. N. d. (Gift of Consul General of Brazil.)

RECONNAISSANCE OF THE GEOLOGY AND OIL PROSPECTS OF NORTHWESTERN OREGON. Bull. 590, U. S. Geological Survey. Washington, 1914.

Assaying, Chemistry, and Physics

DIE HERSTELLUNG DER SPRENGSTOFFE. 1 Teil. By A. Voigt. (Monographien über chemisch technische Fabrikationsmethoden. Band XXXII.) Halle a. S., 1913.

ENZYKLOPÄDIE DER TECHNISCHEN CHEMIE. Vol. 1. By Fritz Ullmann. Berlin-Wien, 1914.

LÖTROHRPROBIERKUNDE. By C. Krug. Berlin, 1914.

MANUFACTURE OF SULPHURIC ACID AND ALKALI WITH THE COLLATERAL BRANCHES. A theoretical and practical treatise. Ed. 4. Vol. 1, pts. 1-3. By George Lunge. New York, 1913.

PHYSICAL PROPERTIES OF THE METAL COBALT. (Canada, Mines Branch, No. 309.) Ottawa, 1914.

TABLES ANNUELLES DE CONSTANTES ET DONNÉES NUMÉRIQUES DE CHIMIE, DE PHYSIQUE ET DE TECHNOLOGIE. Vol. III, 1912. Paris, 1914. (Gift of University of Chicago Press.)

[NOTE.—The third issue of these extremely useful tables of constants, issued internationally.—W. P. C.]

Mineral and Metal Production

DIE MINERALREICHTÜMER UND DIE BERGBAULICHEN VERHÄLTNISSE ARGENTINIENS. By Bruno Simmersbach. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 147.) Kattowitz, n. d.

ECONOMIC MINERALS, AND MINING INDUSTRIES OF CANADA. Canada, Mines Branch No. 322. Ottawa, 1914.

- ENTWICKLUNG UND BEDEUTUNG DER OBERSCHLESISCHEN EISENINDUSTRIE. By E. Zivier. (Sammlung Berg und Hüttenmännischer Abhandlungen, pt. 144.) Kattowitz, n. d.
- GENERAL SUMMARY OF THE MINERAL PRODUCTION OF CANADA DURING THE CALENDAR YEAR, 1913. Canada, Mines Branch, No. 319. Ottawa, 1914.
- GEOLOGY OF THE HANAGITA-BREMNER REGION, ALASKA. Bull. 576, U. S. Geological Survey. Washington, 1914.
- METAL STATISTICS, 1915. 8th Annual Edition. New York, 1915. (Gift of the American Metal Market Co.)
- MINERAL RESOURCES OF OKLAHOMA AND STATISTICS OF PRODUCTION FROM 1901-1914. Bull. 22, pt. 2, Oklahoma Geological Survey. Norman, 1914.
- PRODUCTION OF COPPER, GOLD, LEAD, NICKEL, SILVER, ZINC, AND OTHER METALS IN CANADA, 1913. Canada, Mines Branch, No. 317. Ottawa, 1914.

General

- BIBLIOGRAPHY OF ARIZONA. By Hector Alliot. Los Angeles, 1914.
- BRITISH STANDARD SPECIFICATION AND SECTIONS OF STEEL FISHPLATES FOR BRITISH STANDARD BULL HEAD RAILWAY RAILS. Engineering Standards Committee. (No. 47, Revised Dec., 1914.) London, 1914.
- FUNDAMENTAL BASIS OF ESTIMATING THE QUALITY OF MATERIALS. (In Russian.) By A. Mitinski. Petrograd, 1914. (Gift of Author.)
- GAS POISONING IN MINING AND OTHER INDUSTRIES. By John Glaister and D. D. Logan. New York, 1914.
- LISTS OF BRITISH STANDARD ROLLED SECTIONS FOR STRUCTURAL PURPOSES. (No. 1. Revised Dec., 1914.) London, 1914.
- PAINTS TO PREVENT ELECTROLYSIS IN CONCRETE STRUCTURES. Bull. 47, Paint Manufacturers' Association of the U. S. Philadelphia, 1915. (Gift of Institute of Industrial Research.)

Company Reports

- MOUNT MORGAN GOLD MINING Co., LTD. Reports and Statements of Accounts for half year ended Nov. 29, 1914. Queensland, 1914. (Gift of B. Magnus.)

Trade Catalogues

- BALDWIN LOCOMOTIVE WORKS, Philadelphia, Pa.
Publication 1525. Baldwin-Westinghouse electric mine locomotives.
Instruction Book 5144. Baldwin-Westinghouse electric mine locomotives.
- HARRISON SAFETY BOILER WORKS, Philadelphia, Pa.
Catalogue 601. Cochrane multiport valves.
Cochrane Engineering Leaflet No. 17. Reducing boiler room costs by heating and softening the feed water. 20 pp.
- NATIONAL TRANSIT Co., Oil City, Pa. Bull. Nos. 1-4, 101, 301. Pumps.
- SUPPLEE-BIDDLE HARDWARE Co., Philadelphia, Pa. Monel metal. Jan., 1915, Feb., 1915.
- WESTINGHOUSE ELECTRIC & MFG. Co., East Pittsburgh, Pa.
Central Station power in coal mines. 15 pp.
Westinghouse motors for mine, mill and smelter service.

Notes on Trade Literature

The Jeffrey Manufacturing Co., Columbus, Ohio, has recently issued a new 48-page Bulletin, No. 147, illustrating and describing the prominent features of its complete line of Swing Hammer Pulverizers, giving full information regarding capacities, speeds, horsepower, general dimensions, etc. More than 1,000 of these machines are now in daily operation reducing limestone, shale, gypsum, clay, coal, coke, ores, tankage, bark, oyster shells, rock for road top dressing, and many other materials. A free copy of this bulletin may be obtained by writing to the home office.

The Chas. A. Schieren Co., belt manufacturer, has installed an exhibit at the Panama-Pacific Exposition showing its products and illustrating by working models various methods of drive. All members of the Institute are invited to visit the booth and consult the company's representative concerning their belting problems.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Feb. 10 to Mar. 10, 1915:

Members

- BANKS, HAROLD PURDY, Min. Engr. 61 Broadway, New York, N. Y.
 BARAGWANATH, JOHN G., Min. Engr., Morococha Mining Co., Morococha, Peru.
 BEESON, JOSEPH JOSIAH, Min. Geol. Stanford University, Cal.
 CALLOWAY, ALFRED W., Pres., Davis Coal & Coke Co., Cumberland, Md.
 DEETRICK, JAMES W., Mgr., Republic Iron & Steel Co., 826 Pennsylvania Ave.,
 Youngstown, Ohio.
 DEVINE, JOHN CENTENNIAL, Asst. Supt., Ray Consolidated Copper Co., Ray, Ariz.
 DOUGALL, RALPH, Min. Engr. P. O. Box 582, Bankhead, Alta., Canada.
 DOUGAN, LEE DEWANE, Mill Supt. Mogul Mining Co., Terry, S. D.
 FELDENHEIMER, ROY, Min. Engr. 702 Main St., Portland, Ore.
 FORBES, JAMES HYDE, Geol. and Hydrographer. 1638 Fourth Ave., Los Angeles, Cal.
 FORMIS, ANDRE, Supt., Bates Iron Co., Iron River, Mich.
 GABY, WALTER E., 528 W. 123d St., New York, N. Y.
 GHEUR, ERNEST, Cons. Engr. Red Deer, Alta., Canada.
 GRADY, WILLIAM HENRY, Chief Mine Inspector. 160 Giles St., Bluefield, W. Va.
 GRANT, LESTER STRICKLAND, Genl. Mgr., Jumper Californian Gold Mining Co.,
 Stent, Tuolumne Co., Cal.
 HASTINGS, JOHN HARTLEY, Chem. Box 166, Burgettstown, Pa.
 IRWIN, EUGENE F., Mgr., Employment Dept., Homestake Mining Co., Lead, S. D.
 JOHNS, ALFRED L., Min. Engr. 207 Bullard Bldg., Los Angeles, Cal.
 JOHNSON, ROSWELL H., Prof. of Oil and Gas Production, 306 State Hall,
 Univ. of Pittsburgh, Pittsburgh, Pa.
 JONES, E. HORTON, Chief Engr., Arizona Copper Co., Clifton, Ariz.
 KAY, DAVID NELSON, Min. Engr. Ray Consolidated Copper Co., Hayden, Ariz.
 LANIER, STERLING SIDNEY, Coal Mine Operator, P. O. Box 625, Birmingham, Ala.
 LANIER, STERLING S., JR., Gen. Mgr., Norton Coal Mining Co., Nortonville, Ky.
 MACDOWELL, CHARLES H., Pres., Armour Fertilizer Works, Union Stock Yards,
 Chicago, Ill.
 MIXNER, ALBERT FREDERICK, Mine Supt., Eleanora Mines, Hillgrove, N. S. W.,
 Australia.
 MURPHY, CHARLES J., Min. Engr., Crow's Nest Pass Coal Co., Ltd.,
 Fernie, B. C., Canada.
 NELSON, THEOPHILUS LEEPER, Mine Supt., Engr., Empire Coal Co., Empire, Ala.
 NEWMAN, W. FRANKLIN, Genl. Mgr., No. 8, The Sanctuary, Westminster,
 London, W., England.
 NISHIHARA, GEORGE SHIKATARO, Min. Geol., Osaka, Japan.
 ORD, ROBERT SCOTT, Genl. Mgr., Corbin Coal & Coke Co., Spokane, Wash.
 PAUL, JAMES W., Min. Engr. U. S. Bureau of Mines, Pittsburgh, Pa.
 PEDDAR, THOMAS CLIFFORD, Mgr., Cia. De Minas De Cobre De Gatico,
 Gatico, Chile.
 PERRY, RAY POTTER, Genl. Mfg. Mgr., Barrett Mfg. Co., 17 Battery Place,
 New York, N. Y.
 PICKETT, GEORGE B., Mining. Ophir, Colo.
 QUINN, MAX F., Engr., Assayer. E. 819 Sharp Ave., Spokane, Wash.
 RABLING, HAROLD, Min. Engr. 4515 Forbes St., Pittsburgh, Pa.
 RAJ, SHIV, Mining. care The Punjab Co-operative Bank, Ltd., Lahore, India.
 RITTER, FERDINAND, Supt. Staso Milling Co., Castleton, Vt.
 ROSENGARTEN, GEORGE D., Mfg. Chem. P. O. Box 1625, Philadelphia, Pa.
 RUSSELL, ALEXANDER G., JR., Agent, Faraday Coal & Coke Co., Tazewell, Va.
 SALISBURY, O. J., Pres., Mgr., Deer Trail Mining Co., 207 Felt Bldg.,
 Salt Lake City, Utah.
 SAUNDERS, WALTER M., Anal. and Cons. Chem., 20 Dewey St., Olneyville Sta.,
 Providence, R. I.
 SCHULZE, WILLIAM JOSEPH, Supt. of Mines, Secy., Tesora Mining Co., Virginia, Minn.

STADTMUELLER, KARL C., Concentrator Supt. Latouche, Alaska.
 STOCKETT, THOMAS R., Mgr. Western Fuel Co., Nanaimo, B. C., Canada.
 WATTON, EUGENE E., Treas., Genl. Mgr., United Oil Co.,
 410 First National Bank Bldg., Denver, Colo.
 WETZEL, WALTER N., Mine Supt. Sunnyside, Utah.

Associate Members

JONES, JOHN CHARLES, Branch Mgr., Westinghouse Elec. & Mfg. Co.,
 1212 Walker Bank Bldg., Salt Lake City, Utah.
 MATTHEWS, CHARLES HENRY, Elec. Engr., General Electric Co.,
 801 Fidelity Bldg., Duluth, Minn.
 MONTGOMERY, JOHN C., Mining. 165 Broadway, New York, N. Y.
 ROBERTS, JAMES T., Accountant, Asst. Secy., Anaconda Copper Mining Co.,
 304 N. Main Street, Butte, Mont.

Junior Members

ALINE, PETER A., Student. Montana State School of Mines, Butte, Mont.
 BASSETT, THOMAS E., Student. 332 Channing Ave., Palo Alto, Cal.
 BAYLESS, GEORGE EDWARD SILVER, Student. 2221 St. Paul St., Baltimore, Md.
 GRANT, BERTRAM. Silver Bullion Mine, Brownell, Pima Co., Ariz.
 JOHNSON, JOHN BERNARD. 3513 Eoff St., Wheeling, W. Va.
 LUERSEN, GEORGE V., Student. Penn. State College, State College, Pa.
 REBER, BERTRAM A., Student. Montana State School of Mines, Butte, Mont.
 RHODES, ALBERT E., Student. Penn. State College, State College, Pa.
 WOODWORTH, GUY THOMAS, Student. Montana State School of Mines, Butte, Mont.

Changes of Status—Member to Honorary Member

KEMP, JAMES F., Prof. of Geol. Columbia Univ., New York, N. Y.

Junior Member to Member

DOBSON, C. G., Asst. Engr., Original Mine, 302 Colonial Blk., Butte, Mont.

CANDIDATES FOR MEMBERSHIP

The following persons have been proposed during the period Feb. 10 to Mar. 10, 1915, for election as members of the Institute. Their names are published for the information of members and associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Simon Harry Ash, Seattle, Wash.
 Proposed by George Watkin Evans, Milnor Roberts, Livingston Wernecke.
 Born 1886, Roslyn, Wash. 1903, Roslyn public schools. 1907, Grad., I. C. S., surveying and mapping course. 1909-10, Grad., Bethlehem Preparatory School, Bethlehem, Pa. 1910-12, Lehigh Univ.; E. M. 1914, Grad., International Correspondence School; M. E. 1903-06, Rodman and transitman, mine corps, Northwestern Improvement Co., Tacoma, in mines of Washington and Montana. 1908-09, Chief Draftsman, Northwestern Improvement Co. 1909, Rodman, Pittston Div., Lehigh Valley Coal Co. 1911, Laborer, construction, Northwestern Improvement Co., Roslyn, Wash. 1912, Min. Engr., Roslyn Cascade Coal Co., Roslyn, Wash. 1913, Min. Engr., American Coal Co., Spiketon, Wash.
 Present position: 1913 to date, Deputy State Inspector of Coal Mines.

C. Kemble Baldwin, Chicago, Ill.

Proposed by H. Eckfeldt, Joseph W. Richards, Henry S. Drinker.

Born 1873, Philadelphia, Pa. 1895, Lehigh University; M. E. 1896-99, Drafting, Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1899, Mech. designer, Crocker-Wheeler Co., Ampere, N. J.

Present position: 1899 to date, Chief Engr. and Mgr. of Western office, Robins Conveying Belt Co.

R. E. Burg, Birmingham, Ala.

Proposed by Priestley Toulmin, Erskine Ramsay, Henry S. Geisner.

Born 1886, Ann Arbor, Mich. 1911, Univ. of Mich.; B. S. 1911-13, Div. Engr., Republic Iron & Steel Co., Sayreton, Ala. 1913-14, Draftsman, American Steel & Wire Co., Fairfield, Ala.

Present position: 1914 to date, Min. Engr., Motley & Dryer.

Richard Butler, Ulverston, Lancashire, England.

Proposed by F. W. Linck, Charles F. Rand, John A. Agnew.

Born 1884, Barrow-in-Furness, England. 1903-08, Apprenticeship to Ed. Tachsen, C. & M. E., Whitehaven, England. General training in mining, surveying, etc. Technical training, Whitehaven Collieries. 1906-07, Asst. Engr. to George Boyd, Whitehaven. 1907 to date, studied geology and metallurgy under George Grace, at technical school, Barrow-in-Furness. 1907 to date, entered service of Harrison Ainslie & Co., Ltd., as mine surveyor. 1908, Deputy Underground Manager, over four mines. 1911, Deputy Mine Supt., over seven of the company's mines (hematite iron ore). 1913, Mine Supt., over all the mines. Certificates for mining and surveying, metallurgy, iron and steel manufacture, geology, etc. 1903-08, Mine surveying, hematite iron ore mines, Cumberland. 1908-09, Asst. Engr. to George Boyd, Whitehaven. Engaged in general surveying, railway, waterworks, and sewage.

Present position: 1907 to date, Mines Supt., under Harrison Ainslie & Co., Ltd., Lindal Moor mines.

Dudley S. Dean, Boston, Mass.

Proposed by H. L. Smyth, Charles H. White, George S. Raymer.

Born, 1871, Lake Village, N. H. 1877-84, Public schools. 1884-87, St. Paul's School, Concord, N. H. 1887-91, Harvard Univ.; A. B. 1891-98, A., T. & S. F. Ry. operating department, Iowa, Missouri and New Mexico, four years last-named State. 1898-99, Served in Cuba, Spanish war. Keystone churn drilling gold-dredging ground, Idaho and No. California. In employ of Boston & Idaho Gold Dredging Co., C. H. Souther (since deceased, along with company), President. 1899-1901, Gold dredging, Idaho (Boise Basin), and prospecting and building gold dredge, Quesnelle River Dist. of Cariboo Dist., B. C. Also employed gold-quartz property near Gazelle, Siskiyou Co., Cal. 1901-05, Mercantile work.

Present position: 1905 to date, Treas., Keweenaw Land Assn., Ltd., owning stumpage leads and about 25 active iron mines in upper Michigan.

Howard W. Fitch, Eureka, Utah.

Proposed by Walter Fitch, G. W. Crane, J. R. Finlay.

Born 1881, Marquette, Mich. 1895-98, High School, Beacon, Mich. 1899-1900, St. John's College, Fordham, N. Y. 1901, Asst. chemist and surveyor, Champion mine, Beacon, Mich. 1902, Engineering Dept., Cleveland-Cliffs Iron Co., Ishpeming, Mich. 1903-04, Student, Mich. College of Mines, Houghton, Mich. 1905-06, Engineering Dept., Highland Boy mine, Bingham, Utah. 1907, Mining, Yerington, Nev. 1908, Mining, Rogers-Brown Ore Co., Deerwood, Minn.

Present position: 1909 to date, Asst. to Geol., Chief Cons. Mining Co.

Ernest L. Godbe, Salt Lake City, Utah.

Proposed by H. C. Goodrich, R. C. Gemmell, C. W. Whitley.

Born 1867, Salt Lake City, Utah. 1884-90, studied chemistry and metallurgy, under E. H. Russell and others. Since latter date been actively engaged in practical metallurgical and mining work. Constructed and operated mills at Pioche, Delamar, Nev., Mojave, Cal., Pfau, Ariz., Sevier mill, Utah, Cutting mill, Utah.

Present position: Genl. Mgr., Prince Cons. M. S. Co.

Elmer Grant Goldshalk, Collinsville, Okla.

Proposed by J. A. Van Mater, H. Eckfeldt, Lewis G. Rowand, George C. Stone.

Born 1875, Kulpesville, Pa. 1895, Lehigh Univ., Analytical Chemist. 1896-1901, Chemist and mill supt., New Jersey Zinc Co., Franklin Furnace, N. J. 1901-02, Mill supt., Witherbee, Sherman & Co., Mineville, N. Y. 1902-06, Supt., Columbia Lead Co., Flat River, Mo.

Present position: 1906 to date, Supt., Bartlesville Zinc Co.

Henry J. Gray, Denver, Colo.

Proposed by F. Lynwood Garrison, Frank Bulkley, James M. McClave.

Born 1859, Chicago, Ill. 1876, High School; 1880, also night school, Cleveland, Ohio. 1880-83, Prospecting, Colo., Utah and Ariz. 1884-85, North Star mine, White Pine, Colo. 1885-86, Gunnison Co., Colo.; San Juan, Colo. 1886-87, Leasing, Leadville, Red Cliff, Colo. 1887-1903, A. M. & S., also leasing, Aspen, Colo. 1903-05, Oaxaca, Mexico, Central America. 1905-07, Supt., Robinson mine, Robinson, Colo. 1907-13, Supt., Marion Mines & Mills Co., Denver, Colo. 1913-14, Supt., Excelsior Cons. Gold Min. Co., Cisco, Cal.

Present position: Genl. Mgr., Excelsior Cons. Gold Mining Co., Cisco, Cal.

Wendell C. Hammon, San Francisco, Cal.

Proposed by James C. Ray, G. H. Clevenger, H. W. Young.

Born 1890, Oakland, Cal. 1913, Stanford Univ.; A. B.

Present position: With W. P. Hammon.

Benjamin P. Hazeltine, Wheeling, W. Va.

Proposed by A. N. Diehl, William A. Cornelius, J. W. Wilson.

Born 1877, San Francisco, Cal. 1899, Grad., Mass. Inst. of Tech.; B. S. Elec. Engrg. For seven years in drafting office and mechanical department, National Works, National Tube Co.

Present position: Supt. Bessemer Dept. and Rolling Mills, Riverside Works, National Tube Co.

Lanatt Tinsley Higgins, Toronto, Canada.

Proposed by George A. Guess, Frank E. Lathe, H. E. T. Haultain.

Born 1893, Toronto. 1906, Public schools, Toronto. 1911, High school, University of Toronto, Faculty of Applied Science. 1911, Bell Telephone Co. of Canada. 1914, Mond Nickel Co., Garson Mine.

Present position: Student.

Stephen Adair Holman, Jr., Coram, Shasta Co., Cal.

Proposed by Frank M. Leland, G. W. Metcalfe, J. H. Kervin.

Born 1886, La Graciosa, Cal. 1892-1903, Public schools of Cal. 1903-05, Night study with my father. 1912, Course in surveying at Van der Naillen School of Engineering, Oakland, Cal. Supplemented by private study of technical subjects while employed during the past ten years. 1903, Underground work, Gwin mine, Calaveras Co., Cal. 1904, Underground work in Gwin, Kennedy and other mines in Mother Lode district, Cal. 1905, Underground work in Tonopah, Nev., and Shasta Co., Cal. 1906, Underground work at Mammoth mine, Shasta Co., Cal. 1907, Shift boss, Mammoth mine. 1908, Foreman, Buchanan Copper Mine, Madera Co., Cal. 1909, Office work, Mammoth mine. 1910, Supt., Buchanan copper mine. 1911, Supt., Banner mine, Kingman, Ariz. 1912, Supt., Banner mine, three months school, Oakland, Cal., four months as Mine Supt., Goldroad mine, Goldroad, Ariz. 1913, Mine Supt., Goldroad mine; Asst. Foreman, Mammoth mine.

Present position: 1914 to date, Mine Supt., Balaklala Consolidated Copper Co.

Leon Stuart Holstein, Palmerton, Pa.

Proposed by V. Carl Grubnau, Albert D. Bryant, Lewis H. Haupt.

Born 1890, Akron, Ohio. 1897-1906, Public schools, Akron, Ohio. 1906-08, Public schools, Cleveland, Ohio. 1908-10, Case School of Applied Science. 1910-12, Univ. of Penn.; B. S. 1908-10, Inspector, F. J. Peck & Co., Cleveland, Ohio.

Present position: 1912 to date, Chemist, New Jersey Zinc Co. (of Pa.).

Henry Phelps Howland, Chicago, Ill.

Proposed by H. A. Brassert, George E. Rose, Walther Mathesius.

Born 1878, Marion, Mass. 1908, Drury College, Springfield, Mo.; A. B. 1903, Univ. of Wisconsin, Madison, Wis.; B. S. in M. E. 1903-09, Illinois Steel Co., Joliet-Union and S. Chicago Plants, Blast Furnace departments.

Present position: Supt. Blast Furnaces, Wisconsin Steel Co.

John G. Kirchen, Tonopah, Nev.

Proposed by Horace V. Winchell, Herbert Haas, Warren V. Richardson.

Born 1874, Lake Linden, Mich. 1894, Grad., Michigan College of Mines; E. M. 1894, Asst. Surveyor, Quincy Mining Co., Hancock, Mich. 1898, Surveyor, Arcadian Copper Co., Hancock, Mich. 1901, Gen. Mgr., Shannon Copper Co., Clifton, Ariz. 1908, Gen. Mgr., Montgomery Shoshone Cons. Min. Co.

Present position: Mine Mgr., Tonopah Extension Min. Co.

Ernest Klepetko, New York, N. Y.

Proposed by C. W. Goodale, Joseph Struthers, Frank Klepetko.

Born 1888, Osceola, Mich. 1906-08, Columbia College, N. Y. C. 1908-11, Michigan College of Mines; B. S. 1913-14, Special work in electro-metallurgy and electro-chemistry at Columbia Univ. 1911, Asst. Chem., Lake Superior Smelting Wks. 1911-13, In charge of prospecting and development work of mines of A. C. Burrage and Frank Klepetko in Peru, S. A.

Present position: With F. Klepetko.

Oliver Curtis Martin, Richmond Hill, N. Y.

Proposed by James B. Herreshoff, Jr., Franklin Guiterman, P. A. Mosman.

Born 1873, Rockville, Ind. 1896-1900, State Univ. of Indiana, chemistry as major subject; A. B. 1901-04, American Smelting & Ref. Co.

Present position: Head of Anode and Wire Bar Dept., Nichols Copper Co.

William F. Meredith, New York, N. Y.

Proposed by Bradley Stoughton, W. M. Corse, William H. Bassett.

Born 1873, Orange, N. J. 1894, Princeton; A. B.

Present position: Pres., Titanium Alloy Mfg. Co.

Elmer Randall Ramsey, Denver, Colo.

Proposed by John V. N. Dorr, M. S. McCarthy, George E. Collins.

Born 1888, London, Ky. 1908, Grad., Denver High School. 1912, Colo. School of Mines; E. M. 1905-06, Concentrator mill man, Dives Pelican and Seven Thirty Mining Co., Silver Plume, Colo. 1907-08, Mine and mill, Silver Plume, Colo. 1909, Asst. assay office and sampler at smelter, Lodi, Nev. 1910, Concentrator millman, Wellington mines, Breckenridge, Colo. 1912, mining, Breckenridge, Colo. 1912, Draftsman, Dorr Cyanide Machinery Co., Denver, Colo. 1913, Experimental, Lundberg-Dorr and Wilson Mill, Terry, S. D. 1913-14, Construction, Mogul Mining Co., Terry, S. D.

Present position: 1914 to date, Asst. Engr., Dorr Cyanide Machinery Co.

John Randall, Placerville, Idaho.

Proposed by J. V. N. Dorr, Charles A. Chase, H. N. Spicer.

Born 1853, Dayton, N. Y. High school and correspondence school, but mostly by study out of school. Since 1897 have been engaged in mill construction and ore treatment in the West. Since 1901 mostly cyanide work. Have done a little technical writing. Ex. *Mines and Minerals* (Scranton) 1912-13. 1902, Introduced continuous counter-current decantation in U. S., about the time Denny worked on it in S. Africa.

Present position: Mill Supt., National Min. & Development Co.

Avery H. Reed, Marion, Ky.

Proposed by J. B. Dilworth, Thomas L. Watson, W. R. Ingalls.

Born 1874, Paducah, Ky. 1894-96, 1897-1901, Student, Univ. of Virginia. 1901-02, Chemist, Hillman Land & Iron Co., St. Louis, Mo. 1902, Min. Engr., and R. R. Engr., Hillman Land & Iron Co., St. Louis, Mo. 1902-04, Engaged in special study and field examination of W. Ky. coal field, for metallurgical coking coal. 1904-05, Genl. Supt., Ky. Fluorspar Co., Marion, Ky. 1905, Mgr., Senator Mining Co., Louisville, Ky. 1905-06, Operative Engr., Marion Zinc Co., Marion, Ky. 1906-07, Constructive Engr., W. Ky. Coal Co., Sturgis, Ky. 1907-08, Asst. Engr., Low Moor Iron Co., Low Moor, Va. 1908-09, Mine Supt., Longdale Iron Co., Longdale, Va. 1909, Operative Engr., Independent Phosphate Co., Columbia, Tenn. 1909-1 Supt., Rosiclare Lead & Fluorspar Mines, Rosiclare, Ill. 1911-13, Operative Engr., Aluminum Ore Co., East St. Louis, Ill.

Present position: 1913 to date, Mine operator, and consulting mining engineer, firm of Reed & Wilson.

Ben Gabriel Slaughter, Jr., Copper Cliff, Ont., Canada.

Proposed by J. L. Agnew, J. C. Nicholls, G. E. Silvester.

Born 1880, Tennessee. 1899-1902, Vanderbilt Univ., special engineering. 1895-99, Prep. School (Webb School and B. & H. School). Previous to 1904, varied experience in drafting room and shops of various steel and blast-furnace plants. 1904-06, Chief Engr. and Asst. Mgr., Walsh & Weidner Boiler Co. 1906-12, one year, Chief Draftsman, three years, Asst. Chief Engr. and Supt. Construction, balance of time Chief Engr., Tennessee Copper Co. 1912-13, Associated with Utley Wedge in consulting engineering.

Present position: 1913 to date, Chief Engr., Canadian Copper Co.

Thomas E. Sturtevant, Dover, N. J.

Proposed by Benjamin Nicolls, R. M. Catlin, Frederick A. Canfield.

Born 1854, Rockaway, N. J. 1862-72, Common school practice and home study. 1872-1900, Apprentice, journeyman, foreman, and superintendent, Morris County Machine & Iron Co., Dover, N. J. 1900-15, Supt. and Chief Engr., McKiernan Drill Co. and their successors, the McKiernan-Terry Drill Co. With both companies I have been engaged in building, installing, and designing mining machinery and appliances.

Present position: Chief Engineer, McKiernan-Terry Drill Co.

Richard B. Thomas, Colville, Wash.

Proposed by L. K. Armstrong, Roy H. Clarke, F. C. Bailey.

Born 1865, Wilmington, Del. 1883, Preparatory college at Media, Pa.

Present position: 1887 to date, civil engineering work and mine surveying, U. S. Mineral Surveyor. Three terms as County Engineer, Stevens County.

Allan Griggs Waite, McGill, Nev.

Proposed by Robert H. Richards, C. B. Lakenan, R. E. H. Pomeroy.

Born 1890, Brookline, Mass. 1907-11, Harvard Univ., specializing in geology and mineralogy; A. B. 1911-13, Mass. Institute of Technology. 1911, General underground work, timbering and mining, Calumet & Hecla Mining Co., Calumet, Mich. 1911, General mill operation at Magna mill, Utah Copper Co., Garfield, Utah. 1912-13, Sampling and special test work on concentrating tables, Ohio Copper Co., Lark, Utah. 1913, Metallurgical work in concentrator, Nevada Cons. Copper Co., McGill, Nev.

Present position: Metallurgical report and experimental work.

Harry Munson Warren, Scranton, Pa.

Proposed by R. V. Norris, J. M. Humphrey, Douglas Bunting.

Born 1875, Worcester, Mass. Common schools and high school, Worcester, Mass. 1896, Grad., Worcester Polytechnic Inst.; B. S. 1897, Post graduate, W. P. I.; M. S. 1905, W. P. I.; E. E. 1898-99, Elect. construction work (personal). 1899-1900, Testing Dept., General Electric Co., Schenectady, N. Y.

Present position: 1900 to date, Elect. Engr., Delaware, Lackawanna & Western R. R. Co., including coal mining dept.

Alfred Russell Whitman, Schumacher, Ont., Canada.

Proposed by H. L. Smyth, Alfred C. Lane, R. R. Freeman, Jr.

Born 1882, Oakland, Cal. 1911, Univ. of Calif.; B. S. 1913, M. S. 1906-07, Head Chem. (analytical), C. M. Fassett Co., Spokane, Wash. 1907, Head Chem., Helena Frisco Min. Co., Gem, Idaho. 1907, Green Mt. mine, Howardsville, Colo. 1911-12, Asst. Geol., Southern Pacific Co., Calif. 1912-13, Asst. Mineralogist and Geol., Univ. of Calif.

Present position: Geol., McIntyre Porcupine Mines, Ltd.

Granville Mercer Williams, Brooklyn, N. Y.

Proposed by George C. Stone, J. A. Van Mater, A. D. Beers.

Born 1889, Utica, N. Y. 1907-11, Columbia Univ. School of Mines; M. E. 1911-14, Metallographer, Research Dept., New Jersey Zinc Co., Palmerton, Pa.

Present position: Asst. to Gilbert Rigg, Sales Engr., New Jersey Zinc Co.

Frank Lake Wilson, Berkeley, Cal.

Proposed by S. E. Bretherton, Wilbur E. Henry, Abbot A. Hanks.

Born 1885, Salt Lake City, Utah. 1904, Spokane High School, Wash. 1908-12, Univ. of Cal., Berkeley, Cal.; B. S. During vacations worked in mines, assay office of Balaklala Copper Co., Coram, Cal., surveying, etc. 1904, Asst. Chemist for Val Verde Copper Co., Val Verde, Ariz. 1905, Asst. Accountant, Clarkville Coal Co., New Mexico. 1905, Asst. Chemist, Great Western Gold Co., Ingot, Cal. 1907-08, Head Chemist. 1912-14, Met. Chem., Afterthought Copper Co., Ingot, Cal.

Present position: 1914 to date, Engr. and Accountant, Atkins, Krall & Co., San Francisco, Cal.

Conrad Wolfe, Spokane, Wash.

Proposed by J. C. Haas, L. K. Armstrong, Roy H. Clarke, F. C. Bailey.

Born 1877, South Dakota. Common school and self education. 1897-1902, Mgr., Monarch mine. 1902-06, Mining in California, Nevada, and Idaho.

Present position: 1906 to date, Pres. and Genl. Mgr., United Copper Mining Co.

*Associate Members***Harry James Cantwell, St. Louis, Mo.**

Proposed by Arthur Thacher, Philip N. Moore, H. A. Wheeler.

Born 1859, Sonman, Pa. Common schools. 1888, Washington Univ., St. Louis, Mo.; LL. B. 1890-96, Pres., Central Lead Co. 1896-1901, Pres., Columbia Lead Co. 1898-1902, Pres., Catherine Lead Co.

Present position: In practice of law.

William Elmer Sands, Rye Valley, Baker Co., Ore.

Proposed by W. M. Dake, Jr., L. A. Walker, Howard S. Lee.

Born 1891, Mica, Wash. 1898-1907, Public grade schools, Mica, Wash. 1908-11, Preparatory Department, State College of Washington. 1911-14, Mining course, State College of Wash. 1911, General mill work, Mammoth mill, Wallace, Idaho. 1912, Mucker, trammer, Standard mine, Mace, Idaho. 1913, Crusher, concentrators, etc., Highland mine, Haines, Ore. 1914, Mill construction, Baker Mines Co., Cornucopia, Ore.

Present position: Asst. Assayer, Rainbow mine.

*Junior Members***Daniel Webster Blaylock, Rolla, Mo.**

Proposed by G. H. Cox, C. R. Forbes, D. H. Radcliffe.

Born 1882, Millheim, Mo. 1900-02, Coffeyville Commercial College. 1904-06, Engineering Department, Federal Lead Co., Flat River, Mo. Summers of 1907 and 1908 as above. 1909-13, Engineering department, Madison Coal Corporation, Glencarbon, Ill.

Present position: Student, Mo. School of Mines.

George Walford Cutting, Florence, Colo.

Proposed by William R. Chedsey, Harry J. Wolf, F. W. Traphagen.

Born 1894, Florence, Colo. 1907-11, Florence High School. 1910, State Geological Survey. 1911-12, Florence Auto Co., Florence, Colo. 1912-13, Arkansas Valley Auto Co., Pueblo, Colo. 1911, United States Refining and Reduction Co., Florence, Colo.

Present position: Student, Colorado School of Mines.

Paul Newman Darrington, Ithaca, N. Y.

Proposed by H. Ries, J. S. Hook, R. E. Somers.

Born 1893, Baltimore, Md. 1908-12, Baltimore Polytechnic Institute.

Present position: 1912 to date, Cornell Univ.

William H. Finkseldey, Camden, N. J.

Proposed by W. R. Crane, H. D. Pallister, C. E. McQuigg.

Born 1892, Philadelphia, Pa. 1911, Grad., Camden High and Manual Mining School, N. J.

Present position: Student, Senior Class, Penn. State College.

Hugo George Granting, Florence, Colo.

Proposed by William R. Chedsey, Harry J. Wolf, F. W. Traphagen.

Born 1893, Pueblo, Colo. 1907-11, Florence High School. 1914, Colo. School of Mines. 1910, Oil Well Supply Co., Florence, Colo. 1911-12, Assessment work on claims, Florence, Colo. 1913, Surveying, Dan Woods, Colorado Springs, Colo. 1914, Manhattan M. & M. Co., Manhattan, Nev. 1914, Timekeeper for contractor, Henry Daley, San Diego, Cal.

Present position: Student, Colo. School of Mines.

George W. Greenawalt, Monessen, Pa.

Proposed by W. R. Crane, H. D. Pallister, C. E. McQuigg.

Born 1892, Elizabethtown, Pa. 1911, Grad. High School, Steelton, Pa. 1910, Chemical Laboratory, Penn. Steel Co., Steelton, Pa. 1908-11, Open hearth. 1913-14, Pittsburgh Steel Co., Monessen, Pa.

Present position: Student, Senior in Mining School, Penn. State College.

Clarence M. Hays, Scottdale, Pa.

Proposed by W. R. Crane, H. D. Pallister, C. E. McQuigg.

Born 1894, Scottdale, Pa. 1911, Grad., Scottdale High School. 1911-14, H. C. Frick Coke Co.

Present position: Student, Senior in Mining School, Penn. State College.

Duncan Macmillan Kerr, Northport, N. Y.

Proposed by H. Ries, R. E. Somers, J. S. Hook.

Born 1893, Northport, N. Y. 1907-11, Greenport High School, Greenport, N. Y. 1911, Cornell Univ. Five weeks during the summer of 1914, Anaconda Copper Mining Co., Butte, Mont.

Present position: Student, Cornell Univ.

John C. Loury, Jr., Somerset, Pa.

Proposed by W. R. Crane, H. D. Pallister, C. E. McQuigg.

Born 1892, Somerset, Pa. 1908, Grad., Somerset High School.

Present position: Student, Senior Class, Penn. State College.

Kenneth Charles McCutcheon, Pittsburgh, Pa.

Proposed by H. Ries, J. S. Hook, R. E. Somers.

Born 1892, Pittsburgh, Pa. 1907-11, Pittsburgh High School, Academical course, Pittsburgh, Pa. 1911, Cornell Univ. 1914, Underground work for U. S. Mining Co. in Bingham, Utah. Pittsburgh-Idaho Co., Gilmore, Idaho.

Present position: Student, Cornell Univ.

Walter Randon Maxon, Cortland, N. Y.

Proposed by H. Ries, R. E. Somers, J. S. Hook.

Born 1892, Cortland, N. Y. 1906-11, Cortland Normal School, Cortland, N. Y. 1911, Cornell Univ. 1914, Underground work at U. S. mine, Bingham, Utah. Milling work at Wilbert mine, Wilbert, Idaho.

Present position: Student, Cornell Univ.

Harper Neeld, Reno, Nev.

Proposed by Francis Church Lincoln, Dwight B. Huntley, Herman Davis.

Born 1889, San Antonio, Tex. 1911, Entered University of Nevada.

Present position: Student, Univ. of Nevada.

Nicolaus H. Raiber, Bellevue, Pa.

Proposed by W. R. Crane, H. D. Pallister, C. E. McQuigg.

Born 1890, Allegheny, Pa. 1907-11, Apprentice in drawing room and draftsman, American Bridge Co., Ambridge, Pa.

Present position: Student, Penn. State College, School of Mines.

Owen M. Roberts, Slatington, Pa.

Proposed by W. R. Crane, H. D. Pallister, C. E. McQuigg.

Born 1889, Slatington, Pa. 1911, Grad. preparatory school. 1911-15, Penn. State College. 1905-08, Slate quarry, M. J. Roberts & Co.

Present position: Senior, School of Mines, Penn. State College.

Carl Godfried Stifel, Rolla, Mo.

Proposed by H. A. Buehler, Durward Copeland, G. H. Cox.

Born 1894, St. Louis, Mo. 1907-11, Yeatman High School, St. Louis, Mo. 1911-13, Washington University.

Present position: Student, Mo. School of Mines and Metallurgy.

Change of Status—Junior Member to Member

Roy Leach, Melones, Cal.

Proposed by J. A. Fulton, Mark L. Latham, Ernest H. Simonds.

Born 1882, Rohnerville, Cal. 1902, Cal. Grammar School, Rohnerville, Cal. 1903-05, Univ. of Pacific Academy, San José, Cal. 1905-06, San José High School, San José, Cal. 1908-09, Western School of Commerce, Stockton, Cal. 1909-13, Univ. of Cal., College of Mines; B. S. 1913, Sampler for Simonds & Latham, Melones, Cal. 1913-14, Supt. of Cyanide Plant for reduction of pyritic concentrates, Cons. Pioneer Gold Mines Co., Towle, Cal.

Present position: 1914 to date, Supt. of Refinery, Simonds & Latham.

CHANGES OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Feb. 10 to Mar. 10, 1915. This list, together with the list published in *Bulletin* Nos. 88 to 99, April, 1914, to Mar., 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1914, and brings it up to the date of Mar. 10, 1915.

ARNOLD, LEO F.	958 Montana St., Chicago, Ill.
ATKINSON, LOUIS W., Min. Engr.	Mgr., St. Clair Colliery Co., Eagle, W. Va.
BAIRD, GEORGE A.	1416 Westminster Bldg., Chicago, Ill.
BAKER, HAMILTON W.	Baker Mines Co., Cornucopia, Ore.
BANKS, JOHN H.	61 Broadway, New York, N. Y.
BARBOUR, PERCY E., <i>Care Engineering and Mining Journal</i> ,	
	10th Ave. and 36th St., New York, N. Y.
BATCHELLER, J. H.	P. O. Box 36, Mattapoisett, Mass.
BAYER, A. H. VON,	Supt., Semet-Solvay Co., Lebanon, Pa.
RIGELOW, BRAXTON	American Ambulance, Boulevard d'Guberman,
	Neuilly sur Seine, France.
BLAIR, GEORGE S.	Central Hotel, R. D. 1, East Brady, Pa.
BLANDY, S. H. B.	Ivythorne Manor, "Street," Somerset, England.
BOLLES, M. N.	413 Wall St., Joplin, Mo.
BOYER, S. L.	West 2808 Boone Ave., Spokane, Wash.
BROCK, R. W., Dean, College of Applied Science, University of British Columbia,	
	Vancouver, B. C., Canada.
BROWN, ARTHUR H.	Dome Lake Min. & Mill. Co., South Porcupine, Ont., Canada.
BRULE, F. J., Constr. Engr., Anaconda Copper Mining Co., Great Falls, Mont.	
BURGREN, A. W.	900 61st St., Oakland, Cal.
BURT, CURTIS F.	Pittsburg-Dolores Min. Co., Rockland, via Yerington, Nev.
CARPENTER, EDWIN L.	822 Kearns Building, Salt Lake City, Utah.
CARR, PERCY E. O.	Care Groos National Bank, San Antonio, Tex.
CARTER, EWING	South Pittsburg, Tenn.
CHADBOURNE, H. W.	Ashland, Mass.
CHALMERS, GEORGE, Villa Nova de Lima, via Raposos, E. F. C. B.,	
	Minas Geraes, Brazil.
CHEVRILLON, LOUIS	6 Quai De Cilly, Paris, France.
CLARK, ARTHUR B.	Sheridan, Mont.
COLE, ROBERT J.	McKay Apartments, 7th and Pike Sts., Seattle, Wash.
CONANT, H. D.	Hubbell, Mich.
CUNNINGHAM, NOEL	P. O. Box 44, Douglas, Ariz.
DAKE, W. M., JR.	833 Park Ave., Hot Springs, Ark.
DINGWALL, W. B. A.	136 East Magnolia Ave., San Antonio, Tex.
DRUMHELLER, D. M., JR.	Republic, Wash.
DUNHAM, L. A.	Care Engineers' Club, 32 West 40th St., New York, N. Y.
ESELESTYN, J. N.	Hematite, Ky.
FARRAR, B. L.	Newman Investment Co., El Paso, Texas.
GALLOWAY, A. D. R.	Prestea Block "A," Prestea, Gold Coast, West Africa.
GARD, IRVING R.	Care Park National Bank, Newark, Ohio.
GORDON, ADAM R., Mgr., New York & Honduras Rosario Min. Co.,	
	San Juancito, Honduras.

- GRAUPNER, M. F. General Delivery, San Francisco, Cal.
 HACKETT, WILLIAM H. Raymondville, Tex.
 HACKWOOD, A. W. 432 S. Jackson St., Butte, Mont.
 HEATHCOTE, C. F., Care Institution of Mining and Metallurgy,
 No. 1, Finsbury Circus, London, E. C., England.
 HEGELER, JULIUS W. 2318 N. Vermillion St., Danville, Ill.
 HERIOT, E. MACKAY. Rio Tinto, Prov. de Huelva, Spain.
 HERR, L. B., JR. 511 Seneca St., South Bethlehem, Pa.
 HILL, A. S. Hecla Mining Co., Box 37, Gem, Idaho.
 HILL, C. R. Brown Palace Hotel, Denver, Colo.
 HINCKLEY, ELMER R., Engr., Société Internationale Forestière et Minière du
 Congo, Kuichassa, Belgian Congo, W. Africa.
 HOCKING, R. O. Stent, Toulumne Co., Cal.
 HOFFMANN, KARL F. 2 Rector St., New York, N. Y.
 IVEY, J. H. Stray Park House, Cambourne, Cornwall, England.
 JORDAN, R. D. Maryland Steel Co., Sparrows Point, Md.
 JUDSON, WILBUR. P. O. Box 323, Lansing, Mich.
 KELLOGG, L. O., Associate Editor, *Engineering Magazine*,
 140 Nassau St., New York, N. Y.
 KINNEY, H. D. Instructed to hold all mail.
 KOEHLER, CARL F. Box 305, Elyria, Ohio.
 KOELLE, CARL, Met. Engr., Compania Metalurgica Mexicana,
 82 Beaver St., New York, N. Y.
 KUANG, YING-CHIEH, Student, Livingston Hall, Columbia Univ., New York, N. Y.
 LELAND, EVERARD. Metcalf, Ariz.
 LINDAU, S. PAUL. Care St. Joseph Lead Co., Herculaneum, Mo.
 McALLISTER, J. E., 503 Standard Bank Bldg.,
 15 King St., West, Toronto, Ont., Canada.
 McCREERY, J. HAROLD. 121 Paulison Ave., Passaic, N. J.
 McDEVITT, JAMES E. 4503 McPherson Ave., St. Louis, Mo.
 McINTOSH, F. K. 812 E. 8th St., Traverse City, Mich.
 MACDONALD, AUGUSTUS. 1124 Fremont Ave., S. Pasadena, Cal.
 MARSHALL, H. T. Operator on flotation, Magna Club, Garfield, Utah.
 MERRY, F. CHARLES. 5 Kenmore Place, Brooklyn, N. Y.
 MESSLER, E. L. Pres., Eureka Fire Brick Works, Mt. Braddock, Pa.
 MONTOULIEU, E. I. 398 17th St., Vedado, Cuba.
 MOONEY, J. D. Care B. F. Goodrich Co., Akron, Ohio.
 MORSE, GEORGE H. Republic, Pa.
 MOXHAM, ARTHUR J. 2 Rector St., New York, N. Y.
 NAWATNY, WILLIAM F. 717 N. Walnut St., Danville, Ill.
 NEILL, CHARLES P. Room 616, Woodward Bldg., Washington, D. C.
 NICHOLSON, FRANCIS. Care Rio Grande Valley Bank & Trust Co., El Paso, Tex.
 NORCROSS, F. S., JR., British Columbia Copper Co., Ltd.,
 Copper Mountain, B. C., Canada.
 ORMSBEE, J. J. Tacoma Smelting Co., Tacoma, Wash.
 PERRY, WILLIAM C. 605 R. A. Long Building, Kansas City, Mo.
 PETERS, RICHARD. 1101 Spruce St., Philadelphia, Pa.
 PETERSON, G. WALFRID. Norrlandsgatan 24, Stockholm, Sweden.
 PRICHARD, WILL A. Room 711, 111 Broadway, New York, N. Y.
 PROUT, JOHN W., JR. Care Mascot Copper Co., Dos Cabezas, Ariz.
 READ, T. T. 420 Market St., San Francisco, Cal.
 RICHARD, B. N. Instructed to hold all mail.
 ROCHE, H. M. Genl. Supt., Mt. Hope Mines, Wharton, N. J.
 SANDERS, J. D. 107 Boston Boulevard, East, Detroit, Mich.
 SAWYER, ARTHUR H., Min. Engr. Alabama Power Co., Birmingham, Ala.
 SCHMALZ, CHARLES H. P. O. Box 326, Billings, Mont.
 SEAL, ALBERT E. 49 Mount Park Road, Ealing, London, W., England.
 SHAW, S. F., Marguerite Apartments, Boulevard and Octavia Sts., El Paso, Tex.
 TARB, R. P., Min. Geol. 721 North L St., Tacoma, Wash.
 THOMAS, D. R., Care Zacatecas Mining & Metallurgical Co., Zacatecas, Mexico.
 THOMPSON, J. FRANK. Mine La Motte Co., Mine La Motte, Mo.
 THOMPSON, WILLIAM, 73 Antrim Mansions, Belsize Park, London, N. W., England.
 TOLMAN, J. D., McIntyre-Porcupine Mines, Ltd.,
 P. O. Box 12, Schumacher, Ont., Canada.
 TSUKAKOSHI, U. Furukawa Mining Co., Yaesu-Cho, Kojimachi-ku, Tokio, Japan.

TYSSOWSKI, JOHN.....	200 Fifth Ave., New York, N. Y.
UREN, W. J., Genl. Mgr., Keweenaw Copper Co. and Phoenix Cons.	
VAN DYKE, BURTON, Builder and Contractor....	812 N. 16th St., Harrisburg, Pa.
WARNER, THOR.....	Care J. Emanuel Johnson, Ripon, Cal.
WEBB, H. H.....	36th floor, 233 Broadway, New York, N. Y.
WEIGEL, W. M.,.....	General Delivery, Orillia, Ont., Canada.
WEINBERG, SEMEN GEORGE, Kirkpitchny Pereouluk No. 1, Petrograd, Russia.	
WHITE, R. T.....	Care Caucasus Copper Co., Alexandrofsky, Caucasus, Russia.
WHITTINGHAM, HAROLD.....	Market Rasen, Lincoln, England.
WILKINSON, JOHN FRANCIS, Min. Engr.....	1016 Webster St., Oakland, Cal.
WILLIAMS, MORRIS.....	907 Commercial Trust Bldg., Philadelphia, Pa.
WILSON, M. O.....	14 Wall Street, New York, N. Y.
WILSON, NATHANIEL.....	"Littlehurst," Berkhamsted, Herts, England.
WRAMPELMEIER, E. L. S.....	P. O. Box 491, Nevada City, Cal.
WRIGHT, CLARK W.....	Care Little River Drainage District, Delta, Mo.
WRIGHT, LOUIS A.....	16 Maple St., Bronxville, N. Y.
YOUNG, C. M.....	Editor, <i>Colliery Engineer</i> , Scranton, Pa.

ADDRESSES OF MEMBERS AND ASSOCIATES WANTED

Name	Last address of Record, from which Mail has been Returned.
BLOW, JOHN J.....	173 Rodney St., Brooklyn, N. Y.
GOEDICKE, CARL.....	Box 535, San Antonio, Texas.
GOODLAND, GILMORE.....	17 Gracechurch St., London, E. C., England.
GORDON-FIREBRACE, W. E.....	812 Salisbury House, London, E. C., England.
HYDE, JAMES M.....	1041 Shattuck Ave., Berkeley, Cal.
JONES, EDWARD B.....	Box 502, Salt Lake City, Utah.
MOCINE, JOHN.....	National Copper Min. Co., Mullan, Idaho.
NEWBERRY, R. W.....	Box 1477, Bisbee, Ariz.
PORTER, ROBERT S.....	Care Fortifications, Culebra, Canal Zone.
REVELL, GEORGE E.....	P. O. Box 132, Nelson, B. C., Canada.
STEEL, DONALD.....	202 Emerson St., Palo Alto, Cal.
WRIGHT, PERCY E.....	Seattle, Wash.

NECROLOGY

The death of the following member was reported to the Secretary's office during the period Feb. 10 to Mar. 10, 1915:

Date of Election	Name	Date of Decease
1909	*Underhill, W. A.....	November 7, 1914.

*Member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

L. W. FRANCIS, *Chairman*, WILLARD S. MORSE, *Vice-Chairman*.
 THOMAS T. READ, *Secretary*, Woolworth Bldg., New York, N. Y.
 P. A. MOSMAN, *Treasurer*.

LOUIS D. HUNTOON.

WILLIAM A. POMEROY.

Boston

Meets first Monday of each winter month.

HENRY L. SMYTH, *Chairman*, ALFRED C. LANE, *Vice-Chairman*.
 AUGUSTUS H. EUSTIS, *Secretary-Treasurer*, 131 State St., Boston, Mass.
 ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

FRANK A. ROSS, *Chairman*, RUSH J. WHITE, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.

I. F. LAUCKS, *Chairman*, J. F. MENZIES, *Vice-Chairman*.
 GLENNVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
 W. C. BUTLER, H. L. MANLEY.

Southern California

THEODORE B. COMSTOCK, *Chairman*, SEELEY W. MUDD, *Vice-Chairman*.
 FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
 A. B. CARPENTER, C. COLCOCK JONES.

Colorado

CHARLES A. CHASE, *Chairman*, S. A. IONIDES, *Vice-Chairman*.
 C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
 FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*, JAMES L. BRUCE, *Vice-Chairman*.
 DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
 FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.

G. HOWELL CLEVINGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
 JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
 F. W. BRADLEY, ANDREW C. LAWSON.

*Pennsylvania Anthracite Section*R. V. NORRIS, *Chairman*.

CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
 EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
 CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

HERBERT A. WHEELER, *Chairman*, FIRMIN V. DESLOGE, *Vice-Chairman*.
 WALTER E. MCCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 JAMES MALCOLMSON, R. A. BULL, PHILIP N. MOORE.

Chicago

ROBERT W. HUNT, *Chairman*, J. A. EDE, *Vice-Chairman*.
 HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.
 F. K. COPELAND, G. M. DAVIDSON.

Utah

R. C. GEMMELL, *Chairman*, C. W. WHITLEY, *Vice-Chairman*.
 ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
 WALTER FITCH, WILLIAM WRAITH.

STANDING COMMITTEES

*Executive*WILLIAM L. SAUNDERS, *Chairman*GEORGE D. BARRON,
SIDNEY J. JENNINGS,JOSEPH W. RICHARDS,
BENJAMIN B. THAYER.*Membership*JOHN H. JANEWAY, *Chairman*KARL EILERS,
LEWIS W. FRANCIS,LOUIS D. HUNTOON,
ARTHUR L. WALKER.*Finance*CHARLES F. RAND, *Chairman*.

GEORGE D. BARRON,

ALBERT R. LEDOUX.

*Library*E. GYBBON SPILSBURY, *Chairman*.⁴KARL EILERS²
JOHN HAYS HAMMOND,¹ALEX C. HUMPHREYS,³
BRADLEY STOUGHTON.*Papers and Publications*BRADLEY STOUGHTON, *Chairman*.

EXECUTIVE COMMITTEE

KARL EILERS,
JOSEPH W. RICHARDS,
JOHN BIRKINBINE,
WILLIAM H. BLAUVELT
H. A. BRASSERT,
WILLIAM CAMPBELL,
R. M. CATLIN,
ALLAN JAY CLARK,
FREDERICK G. COTTRELL,
NATHANIEL H. EMMONS,
JOHN W. FINCH
CHARLES H. FULTON
F. LYNWOOD GARRISON,
ROBERT C. GEMMELL,
CHARLES W. GOODALE,
HARRY A. GUESS,
R. DAWSON HALL,
PHILIP W. HENRY,
HEINRICH O. HOFMAN,WALTER E. HOPPER,
HENRY M. HOWE,
LOUIS D. HUNTOON,
J. E. JOHNSON, JR.,
LEE O. KELLOGG,
WILLIAM KELLY,
JAMES F. KEMP,
CHAS. K. LEITH,
ANTHONY F. LUCAS,
EDWARD P. MATHEWSON,
HERBERT A. MEGRAW,
RICHARD MOLDENKE,
SEELEY W. MUDD,
R. V. NORRIS,
EDWARD W. PARKER,
EDWARD D. PETERS,
ROSSITER W. RAYMOND,GEORGE C. STONE.
SAMUEL A. TAYLOR.
THOMAS T. READ,
ROBERT H. RICHARDS,
L. D. RICKETTS,
HEINRICH RIES,
E. F. ROEBER,
RENO H. SALES,
ALBERT SAUVEUR,
HENRY L. SMYTH,
A. A. STEVENSON
RALPH H. SWEETSER,
SAMUEL A. TAYLOR,
FELIX A. VOGEL
ARTHUR L. WALKER,
ROLLA B. WATSON,
HORACE V. WINCHELL,
FREDERICK W. WOOD,
DWIGHT E. WOODBRIDGE.

COMMITTEE ON JUNIOR MEMBERS AND AFFILIATED STUDENT SOCIETIES

HARRY H. STOEK, *Chairman*.*Vice-Chairmen*CHARLES H. FULTON,
FREDERICK W. SPERR,
WALTER R. CRANE, *Secretary*, Pennsylvania State
LUTHER W. BAHNEY,
DARSIE C. BARD,
ROBERT H. BRADFORD,
SAMUEL W. BEYER,
GUY H. COX,
JOSEPH DANIELS,
NOAH F. DRAKE,WILLIAM B. PHILLIPS,
GEORGE J. YOUNG,
Charles College, State College, Pa.
CHARLES J. NORWOOD,
GEORGE S. RAYMER,
HEINRICH RIES,
HENRY L. SMYTH,
FRANCIS A. THOMSON,
CLINTON M. YOUNG.

COMMITTEE ON ARRANGEMENTS, SAN FRANCISCO MEETING, 1915

E. H. BENJAMIN,
F. W. BRADLEY,C. W. MERRILL, *Chairman*.
ABBOT A. HANKS,H. C. HOOVER,
W. C. RALSTON.¹ Until Feb., 1916.² Until Feb., 1917.³ Until Feb., 1918.⁴ Until Feb., 1919.

COMMITTEE ON INCREASE OF MEMBERSHIP

THOMAS T. READ, *Chairman.*DAVID H. BROWNE, *First Vice-Chairman.**Vice-Chairmen*

LYNDON K. ARMSTRONG,
EDWARD H. BENJAMIN,
DUNCAN MACVICHIE,
VAN H. MANNING,

HENRY W. NICHOLS,
ERSKINE RAMSAY,
SUMNER S. SMITH,
W. G. SWART.

EDWARD L. DUFOURCQ, *Secretary*, 22 Produce Exchange Annex, New York, N. Y.

HUNTINGTON ADAMS,
FRANKLIN BACHE,
EDWIN G. BANKS,
PERCY G. BECKETT,
JAMES G. BERRYHILL,
ALBERT C. BOYLE, JR.
D. H. BRADLEY, JR.
VICTOR M. BRASCHI,
FREDERICK K. BRUNTON,
CHARLES A. BUCK,
MILTON A. CAINE,
EDWIN E. CARPENTER,
CHARLES CATLETT,
NOAH F. DRAKE,
J. A. EDE,
LOUIS V. EMANUEL,
AUGUSTUS H. EUSTIS,
ARTHUR B. FOOTE,
STEPHEN L. GOODALE,
CARL E. GRUNSKY, JR.,

GEORGE A. GUESS,
BENJAMIN M. HALL,
RICHARD S. HASELTINE,
PHILIP W. HENRY,
FRANK R. HEWITT,
JOHN T. HILLES,
JOHN HOATSON,
ROY J. HOLDEN,
CLANCY M. LEWIS,
I. P. LIHME,
JOHN J. LINCOLN,
DOUGLAS C. LIVINGSTON,
SPENCER R. LOGAN,
BRUNO V. NORDBERG,
HENRY M. PARKS,
RICHARD C. PATERSON, JR.
AMBROSE E. RING,
OSCAR ROHN,
WILLIAM W. ROSE,
HAZEL L. SCAIFE,

WILLIAM J. SHARWOOD,
S. F. SHAW,
JO E. SHERIDAN,
ARTHUR P. SILLIMAN,
JOHN G. SMYTH,
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Occurrence of Covellite at Butte, Mont.

BY A. PERRY THOMPSON, BUTTE, MONT.

(San Francisco Meeting, September, 1915)

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I. GENERAL DISTRIBUTION, APPEARANCE, AND RELATIONS

MINING in Butte has seldom encountered covellite in commercial quantities. The notable occurrences, extending vertically and laterally perhaps several hundred feet, presented scattered, bladed lenses of this mineral in a matrix of quartz, pyrite, enargite, bornite, and chalcocite. But under the microscope covellite is seen to be widespread in the Butte ores, though in such small quantity that megascopic examination will not discover it.

In early mining operations covellite was found in the Gray Rock vein, a member of the Blue vein system. As described by Sales,¹ it persisted as an occasional ore mineral, associated with quartz, pyrite, tetrahedrite, bornite, chalcocite, and minor amounts of sphalerite, from the 700-ft. to the 1,200-ft. level of the East Gray Rock and Diamond mines. Crushed quartz-monzonite and later quartz completed the vein filling between

¹ Reno H. Sales: Ore Deposits at Butte, Mont., *Trans.*, xlv, 3 to 109 (1913).

strong clay walls, sometimes as much as 20 ft. apart, while in other places along the strike the vein pinched to a few inches of clay. The mineralization was a replacement of crushed quartz-monzonite between the walls, and formed an ore shoot several hundred feet long, pitching to the SE. and terminating upward, 500 ft. below the surface.

Later, intermittent bunches of covellite were encountered in the Speculator mine, on the Edith May vein, also a member of the Blue vein system. In the vein matter which was capped by barren fault clay and crushed quartz-monzonite, the covellite continued from the 700-ft. to the 1,600-ft. level and laterally several hundred feet. Associated with it in the fault vein were the minerals shown in the Gray Rock vein, with the addition of considerable enargite. Tetrahedrite was found in sparse but universal distribution. The old stopes, in these ore bodies of the Gray Rock and Speculator mines, have long been abandoned and no record has been preserved of their structural or mineralogical details.

Minor amounts of covellite have been found in the veins of the Mountain View, Tramway, Diamond, High Ore, Bell, and Buffalo mines. It appears where conditions in the primary mineralization were favorable for the development of small amounts of the mineral, frequently in crystal form and in vugs, associated with pyrite, enargite, sphalerite, tetrahedrite, bornite, chalcopyrite, and chalcocite. The association of minerals is very similar to that observed with the microscope in the massive ores when the covellite is very small.

In the sooty chalcocite, the presence of covellite is often attested by an indigo-blue tarnish and a bluish powder, frequently present in the enriched portions of all the copper veins. This type of covellite marks an intermediate stage in the enrichment of the primary copper ores and bears no relation to the bodies of primary cupric sulphide below.

In the Leonard mine, large stopes are now worked on the Colusa-Leonard vein, a member of the Anaconda system.² In these stopes, for 600 ft. below the 1,400-ft. level, covellite can occasionally be seen in an area of quartz-monzonite which has been subjected to intense replacement. Here the upper limit of covellite occurs near the 1,000-ft. level, and shreds and stringers of bladed masses in the vein filling have been frequently found along the strike of the orebody.

This unusual development of covellite lies in an area of intense alteration and rich mineralization of the quartz-monzonite. In the portion of the Leonard vein now exposed, where covellite occurs, the "horse-tail" structure of the orebody is well defined. The course of the main mineralization and fissuring is in general E-W.; but all along the vein where covellite has been noted, many seams and veinlets of ore strike off to the south. The whole mass of the country rock, a normal quartz-monzonite,

² R. H. Sales: *loc. cit.*, p. 14.

has been thoroughly shattered south of the north wall of the Colusa-Leonard vein. The main zone of fissuring has been developed along the E-W. orebody. Most commonly the main fissures and minor horse-tail seams, spraying off to the south, are within a few degrees of the vertical, since the strongest movement has taken place in that direction, but the resultant forces produced many minor cracks in every conceivable direction between the larger fracture planes. These cracks are all very irregular and frequently of almost microscopic width. The system of fractures has formed a passageway for mineralizers coming from below and from the main fissure zone and is filled with mixtures and pure veinlets of quartz, pyrite, enargite, covellite, and chalcocite. The quartz-monzonite, generally speaking, is very solid. Veinlets show no displacement or crushing effects. Vugs are often seen in the veinlets and adjacent rock. In the main E-W. fracture zone and between the horse-tail seams and cracks, the quartz-monzonite is sericitized, pyritized, and silicified. Areas of country rock in the zone of intense fracturing have been thoroughly permeated by the early mineralizers. At high temperatures the quartz-monzonite seems to have been comparatively permeable to the solutions causing sericitization, pyritization, and silicification. That the mineralizers continued to obtain access to the solid country rock through the period of ore deposition is evidenced by the abundant replacement of the ferro-magnesian minerals by enargite and by lenses of chalcocite along minute fractures in the solid rock.

Enargite is the most abundant copper mineral in the covellite-bearing areas. It is found replacing the quartz-monzonite in solid veinlets from microscopic size to 3 or 4 ft. in thickness. These larger masses, when broken open, show the enargite well crystallized throughout the entire veinlet. Often smaller stringers in the solid rock have a fine lining of quartz on each side of the enargite filling. Such quartz linings are at times without definite crystal development and again exhibit perfect crystalline forms extending into the enargite, affording evidence that the whole mineralization took place in previously formed cavities. Many enargite veinlets do not have a quartz lining and afford fine examples of replacement of quartz-monzonite along fracture planes. Veinlets several inches thick show no pyrite. Some (always small) cracks are filled with pyrite alone. From the undisturbed appearance of these cracks and replacement seams, it is inferred that there was little movement in parts of the horse-tail area after the original shattering of the country.

Covellite blades, often several inches long, occur in the enargite veinlets, which often show no crushing such as might have afforded an opening for the later deposition of covellite. The cupric sulphide may be present in one foot of a veinlet, while the next foot, in either direction, may pinch out or be pure enargite. In many instances, there is no apparent connection between the masses of covellite. Covellite also occurs in veinlets in



Co, covellite; E, enargite; G, gangue; P, pyrite.
1,600-ft. level, Leonard mine, Leonard vein.

FIG. 1.—COVELLITE STRINGERS THROUGH ENARGITE, PYRITE, AND QUARTZ THAT HAS REPLACED QUARTZ-MONZONITE. V-SHAPED HORSE OF COUNTRY ROCK AT BOTTOM OF VIEW. NATURAL SIZE.

quartz-monzonite, without apparent association with enargite, and in bladed crystals replacing the country rock, sometimes several inches from veinlets of any sort.

In Fig. 1, a stringer of covellite blades may be seen running through enargite, quartz, and pyrite that have replaced the quartz-monzonite. A V-shaped horse of altered country rock may also be seen. The ferromagnesian minerals have been replaced by enargite, chalcocite, and covellite. The quartz and feldspar are still partly preserved. Fine veinlets of enargite, covellite, and chalcocite ramify through the included rock. Residual quartz crystals still remain in the covellite. The cupric sulphide has replaced the quartz-monzonite, enargite, and pyrite.

In the portion of the quartz-monzonite showing the most crushing and alteration, there is no covellite. Here the development of quartz, pyrite, enargite, and chalcocite is greatest. Large amounts of chalcocite replace the previously formed minerals. Near the zone of greatest movement are found many more vugs and a greater development of silica, together with a more marked alteration of the wall rock.

It appears that, after the original fracturing, solutions penetrating the crushed zones, dissolved parts of the quartz-monzonite and to a considerable extent formed solution channels and vugs. The alteration of the country rock was most intense along the E-W. lines of greatest fracturing and in them the greatest silicification occurred. In this zone of most intense fracturing, in the vugs and in the fractures adjacent to the main crushed zone, were deposited first, quartz and pyrite, and probably contemporaneously or nearly so, a large amount of enargite. South of this zone, in the horse-tail area, the quartz-monzonite has been greatly shattered and the cracks have been chiefly filled with, and the country rock replaced by, pyrite and enargite. Quartz is frequently developed as a lining to the enargite veinlets and pyrite occurs sparingly in seams.

From the appearance of these veinlets and vugs, we infer that there was no interval of time, or cessation of mineralization, during the deposition of quartz, pyrite, and enargite. It appears most likely that the mineralizing solutions depositing these minerals entered the shattered quartz-monzonite after its alteration by solutions which caused sericitization and dissolution and deposited their burden in the zone of main fracturing. Quartz and enargite predominated in the early mineralization. Considerable enargite continued to be deposited after the quartz had solidified and had been crushed. Solutions permeated the shattered horse-tail area, depositing enargite after the replacement of the quartz-monzonite with earliest minerals. At this time many of the veinlets were formed in the minor cracks, some of which were due to solution, and in such fashion that a lining of silica was precipitated before the main filling with enargite.

Many stringers of enargite, up to 2 ft. in width, show no chalcocite.

These bodies of enargite frequently contain inclusions of quartz-monzonite and residual quartz, affording conclusive evidence that the stringers involve replacements of the country rock and are not merely filled fissures.

Final solutions ascending from the source of mineralization were very active in dissolving enargite and covellite. Many vugs are found where these two minerals have been attacked by later waters. Enargite particularly was susceptible to the action of the later solvents. Skeletons of enargite crystals are commonly preserved. These vugs universally contain plentiful but small crystals of barite. It is believed that the solvents attacking the enargite and covellite were comparatively alkaline and that the presence of barite, later than the covellite and possibly chalcocite, adds strong evidence to their determination as primary minerals.

The above discussion deals only with that part of the Leonard vein where covellite is found. Other areas of horse-tail ore exhibit a different association of minerals. Much of the enargite of the horse-tail region has been attacked by mineralizers which were depositing chalcocite. Stringers of glance from the finest dimensions up to 2 ft. in width are found, which were precipitated by a direct replacement of solid quartz-monzonite. At times the country rock is peppered with shots of chalcocite, similar to the distribution of enargite described above.

II. DATA FROM MICROSCOPIC EXAMINATION

1. *Criteria for Distinguishing Primary and Secondary Minerals.*

The question of the primary or secondary character of the deep chalcocite orebodies in Butte has long been a matter of argument. Sales³ was the first to furnish conclusive proof, both structural and petrographic, of the primary deposition of chalcocite in all periods of the Butte vein formation. Several investigators, through the microscopic examination of polished sections of the copper ores, have discovered features peculiar to each type of chalcocite. The results promise to make possible the microscopic determination of its primary or secondary nature.

In the study of the bornite and chalcocite ores of the Virgilina district of North Carolina and Virginia, Laney⁴ found graphic intergrowths of bornite and chalcocite resembling the pattern of an eutectic alloy. Similar structures were found in the bornite and chalcocite of the Guilford mine, Guilford County, North Carolina, by Graton and Murdoch.⁵ Eutectic

³ *Loc cit.*, p. 93.

⁴ F. B. Laney: Relation of Bornite and Chalcocite in the Copper Ores of the Virgilina District of North Carolina and Virginia, *Economic Geology*, vol. vi, No. 4, p. 407 (June, 1911).

⁵ L. C. Graton and Joseph Murdoch: The Sulphide Ores of Copper: Some Results of Microscopic Study, *Trans.*, xlv, 76 (1913).

structures in the development of bornite and chalcocite intergrowths have been found in the ore of the Leonard and Jessie veins at Butte.

Bornite in the Leonard vein is associated in small quantities with covellite. In one instance an occurrence of an eutectic pattern of bornite and chalcocite was observed where the alteration of bornite to chalcocite had not progressed far. The eutectic pattern, though seldom observed, gives strong evidence for the primary nature of the chalcocite. The Jessie vein, described by Sales, belonging to the Blue fault system, is unusual in its mineral composition. The oreshoot furnishing the material examined shows a complete transition, between the 500-ft. and 800-ft. levels, from an ore consisting of quartz, pyrite, and chalcopyrite to a mixture of quartz, pyrite, enargite, chalcopyrite, and chalcocite. In ore from this lower zone of mineralization, graphic intergrowths of bornite and chalcocite were found. Bornite, enargite, and chalcopyrite, all undoubtedly primary minerals, contained irregular patches of chalcocite, to all appearances primary, but later than the bornite. This occurrence merges into prominent developments of eutectic structures.

An eutectic alloy consists of an intimate mechanical mixture of two solid phases, produced by crystallization at a constant temperature. The parallel arrangement of the two phases indicates that the crystallization is not simultaneous, but that the molecules of the two phases crystallize alternately. This deposition begins at a number of centers and the interference of the crystal systems leads to the formation of polyhedral masses, composed of an intergrowth of the two phases. The crystallization of bornite and chalcocite in this system of eutectic separation presents strong evidence of the simultaneous deposition of the two minerals and of the primary character of the chalcocite.

Primary intergrowths of bornite and chalcocite have been described by Laney,⁶ and by Graton and Murdoch.⁷

Such an intergrowth is a characteristic mode of occurrence of many of the primary copper minerals of Butte. Enargite and bornite commonly exhibit this structure. Bornite and chalcopyrite are found similarly intergrown. Primary intergrowth presents an irregular interlocking of different minerals, generally without strongly developed crystallization—a meshy, entwined mass of one mineral intergrown with another, preserving at the same time, sharp mineralogical contacts. This primary intergrowth structure is commonly found in eutectic alloys. Eutectics of copper and copper phosphide, or of bismuth and lead, often develop structures identical to that found in the primary intergrowths of the copper ores. As a proof of primary origin of chalcocite, this type is no less conclusive than the more perfect eutectic with parallel arrangement.

⁶ *Economic Geology*, vol. vi, No. 4, p. 406 (June, 1911).

⁷ *Trans.*, xlv, 77 (1913).

different types of replacement may be distinguished in the formation of secondary glance. The first type includes characteristic veinlets of secondary chalcocite ramifying through the primary ore. These veinlets may vary from minute cracks to masses of such size that the original ore entirely disappears. In the development of the cracks certain features are prominent. Almost invariably the fracture from which enrichment progresses is preserved in the center of the veinlet, either as a fracture or as a tabular body of copper minerals or of gangue. Most commonly, the alteration of the primary ore involves the production of intermediate products which form a transition zone between the original ore and the final development, in the veinlet, of chalcocite. This vague cloudy border and the dark medial line are typical of secondary chalcocite. (See Fig. 3.)



C, chalcocite; P, pyrite.
300-ft. level, Silver Bow mine.

FIG. 3.—PYRITE SEAM IN WALL ROCK. SECONDARY CHALCOCITE STRINGERS IN PYRITE. $\times 80$.

A second type of secondary enrichment is found in soft, friable masses of pyrite. Frequently the action of descending sulphuric acid or sulphate solutions tends to render solid pyrite very friable and porous. When deposition of secondary chalcocite takes place in such veins, the action does not proceed from fractures but from many points throughout the entire mass of pyrite. These conditions tend to build up separate grains of sooty glance around many centers, each individual developing a botryoidal mass. Continued action results in lenses of massive glance, more brittle and of much darker luster, generally, than primary glance, with a notable

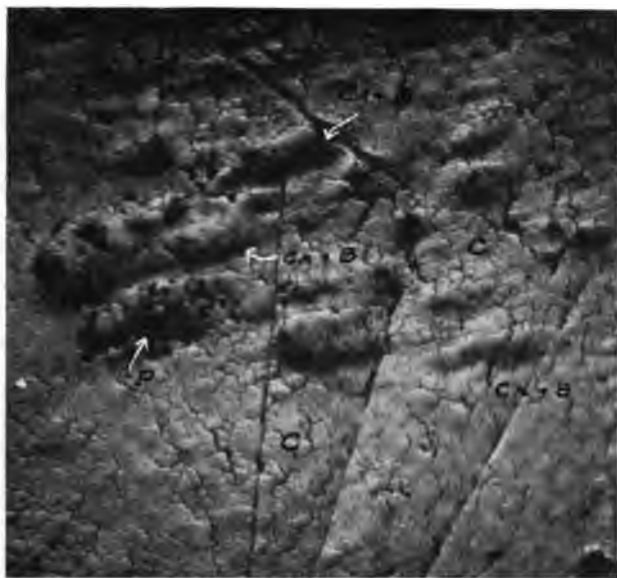
development of botryoidal, stalagmitic structure (Fig. 4). Lenses of such material frequently contain columnar and stalagmitic forms 6 in. long, the



C, chalcocite; P, pyrite.
400-ft. level, Berkeley mine, Snohomish vein.

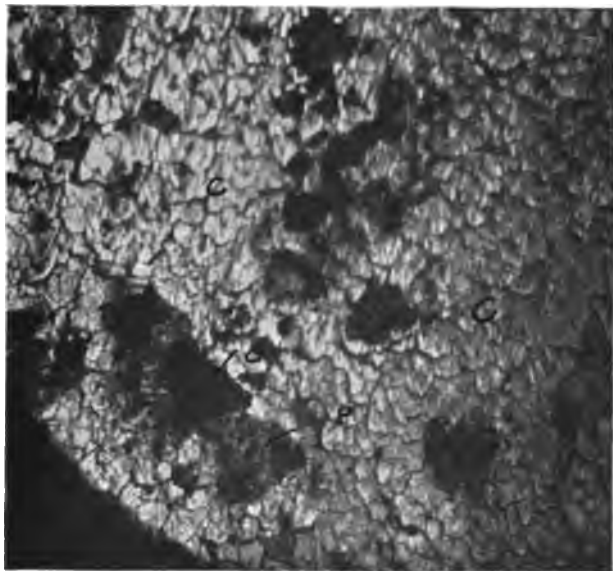
FIG. 4.—BOTRYOIDAL SECONDARY CHALCOCITE WITH REMNANTS OF ORIGINAL PYRITE VEIN FILLING. FULL SIZE.

whole molded together to make a compact ore. In the vicinity of the lenses, all stages of development of the structure may be observed, from the early sooty deposit on pyrite grains to solid botryoidal stalagmites.



B, bornite; C, chalcocite; Ca, chalcopyrite; P, pyrite.
400-ft. level, Berkeley mine, Snohomish vein.

FIG. 5.—BORNITE HALO AROUND PYRITE REMNANTS IN SECONDARY CHALCOCITE. $\times 80$.



C, chalcocite; G, gangue; P, pyrite.
400-ft. level, Berkeley mine, Snohomish vein.

FIG. 6.—SECONDARY CHALCOCITE ETCHING STRUCTURE. $\times 80$.

In the enrichment of pyrite by descending acid, cupriferous solutions, chalcopyrite is first developed as a halo around the pyrite. Bornite next forms, around the chalcopyrite; with the further addition of copper, the final stage is chalcocite. The series furnishes a record of the orderly addition of copper and of the subtraction of iron. Remnants of pyrite surrounded by halos of secondary chalcopyrite and bornite may be seen in Fig. 5.

The above structures have been found to afford etch figures entirely different from those of primary chalcocite. When secondary chalcocite is etched, lines are developed in very ragged fashion. Etching into regular grains is seldom seen. The whole mass presents a disintegrated granular mineral matrix. (See Fig. 6.) Corroborative evidence of the soundness of this structure as a test for secondary chalcocite has been obtained by etching secondary chalcocite from the Briggs, Shattuck, and Junction mines of Bisbee, Ariz. A notable feature of the etching of massive secondary chalcocite is the scaly overlapping of sections between cracks (Fig. 6). This may be called a fish-scale structure. It is due to deeper etching on certain sides of the grains than on others. Such variation in the attack of the acids is believed to be due to the botryoidal texture.

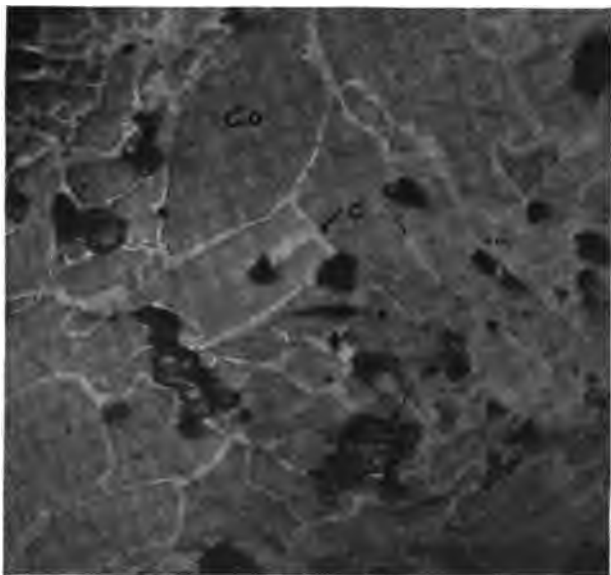
2. *The Leonard Vein Filling*

The copper minerals of the Leonard vein bear a constant relation to one another in form and paragenesis. Examination of many polished sections from the horse-tail ore leads to conclusions concerning the formation of the ore minerals that appear generally applicable.

Evidence in the hand specimens points to the deposition of quartz, pyrite, and enargite as the first stage of mineralization. Quartz was formed first and continued to be deposited after the crystallization of enargite. Similarly, pyrite is found of a later age than enargite. In places, principally along the main E-W. fracture zone, considerable quartz and pyrite were formed before slight shattering admitted cementing solutions of enargite. Examples are plentiful of the replacement of the ferro-magnesian minerals first, and later of all the rock-forming minerals of the quartz-monzonite, by quartz, pyrite, and enargite. Enargite has been particularly active in this respect, filling solution channels, vugs, and fractures.

Quartz.—In polished sections, quartz is seen to have been deposited first, and also, in minor amount, during the successive stages of mineralization. At times good hexagonal crystals, bounded by the copper minerals, are preserved; and again, irregular grains show signs of alteration by the ascending solutions. Quartz, with pyrite, appears to be the material most resistant to alteration. Some quartz remnants in the vein are original crystals from the quartz-monzonite.

Pyrite.—Pyrite in small and slightly shattered grains is peppered through the ore. It is surrounded and replaced by enargite, covellite, chalcocite, bornite, and chalcopyrite. These remnants of former pyrite crystals have been strongly attacked by later solutions. Alteration is evidenced by the corroded rough outlines of the former crystals and by areas of the later copper minerals inclosed in crystals of the iron sulphide. Solutions which deposited covellite and chalcocite have been particularly active in attacking the pyrite. Results of this dissolution show rounded remnants of pyrite grains imbedded in covellite and chalcocite. (See Fig. 7.) Solutions containing enargite have also attacked the pyrite. When enargite, covellite, or chalcocite is found within crystals of pyrite,



C, chalcocite; Co, covellite; P, pyrite.
1,600-ft. level, Leonard mine, Leonard vein.

FIG. 7.—INITIAL ALTERATION OF COVELLITE BY CHALCOCITE. THE MINUTE SPECKS ARE SURVIVING GANGUE. $\times 80$.

frequently no feeding fracture for this mineralization can be observed. It is believed that the mineralizers have gained access to the interior of pyrite grains along lines of weakness or cleavage and have replaced the iron sulphide directly, volume for volume. Many small masses of chalcocite thus formed in large crystals of pyrite, show no indication of bornite or chalcopyrite as an intermediate stage in the alteration.

Enargite.—By reason of its early deposition and abundance, enargite shows the effect of all the later changes in the veins. Little movement is evidenced by the enargite veinlets. It is probable that later movement was largely along the E-W. fractures, leaving the enargite stringers of the

Minute fracture planes, cleavage faces, and cavities between crystals, have united to produce a porous structure in some of the deposits of enargite. Moreover, in the Leonard mine, the deposition of covellite appears to have been limited to enargite areas. Such a selective action is believed to be due chiefly to structural features and local conditions of the mineralizers. In a cross-section of the Leonard vein, bunches of covellite will exhibit a decidedly local arrangement. Much enargite, containing no covellite, has been mined. Country rock in the vicinity of covellite sometimes contains small crystals of the cupric sulphide. Perfectly formed covellite crystals occur in vugs of the veins and of the quartz-monzonite.



C, chalcocite; Co, covellite; E, enargite.
1,600-ft. level, Leonard mine, Leonard vein.

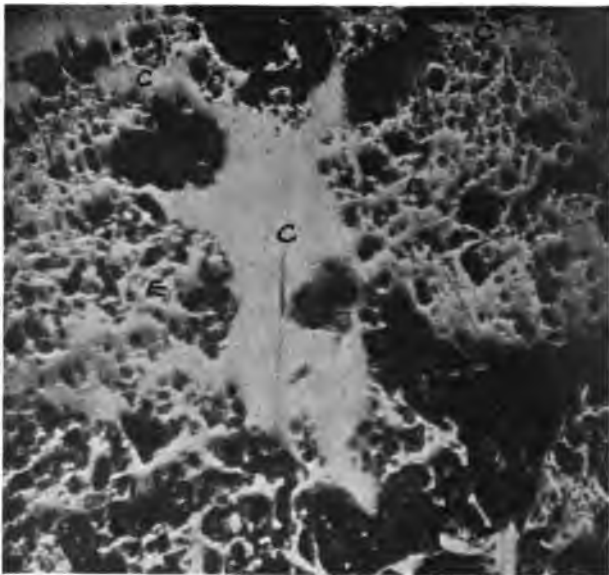
FIG. 9.—REPLACEMENT OF ENARGITE BY COVELLITE AND CHALCOCITE. $\times 80$.

The part that enargite has played, from physical evidence, may thus be said to be that of a favorable medium for replacement and the deposition of other copper minerals, governed by the porosity of the veins and the local character of the mineralizers.

Chalcocite has been very active in the replacement of enargite. With this phenomenon a direct transition from the arsenical sulphide to glance occurs, as illustrated by Fig. 10. Sometimes the transition zone is marked by a bright white rim surrounding the enargite. Again, small areas of enargite in the process of alteration to chalcocite present this appearance. The white material probably represents an intermediate

stage of alteration, either chemical or microscopically physical, between the sulph-arsenate and the cuprous sulphide. With the structure of enargite it displays the color of chalcocite and a peculiar brightness of its own. No such zone is observed in the alteration of enargite to covellite.

Coincident with the replacement of the quartz-monzonite minerals, more or less perfect crystal form has been preserved in enargite. Previously deposited quartz and pyrite have influenced enargite in its crystallization; but the orthorhombic angles and faces adjust themselves to this limitation.



C, chalcocite; E, enargite.
1,600-ft. level, Leonard mine, Leonard vein.

FIG. 10.—REPLACEMENT OF ENARGITE BY CHALCOCITE. $\times 80$.

Covellite.—By far the larger part of the covellite in the Leonard ore-body is a direct replacement of enargite by ascending primary solutions—the upward secondary sulphide enrichment of Tolman⁹ and Rogers.¹⁰ As already shown, the deposition of covellite has been governed by structural, physical, and chemical conditions. The most pronounced development in the Leonard vein has occurred in that part of the E-W. mineralization where the horse-tail stringers begin to spray off to the south. Locally, small lenses and stringers of covellite have been formed for 20

⁹ C. F. Tolman, Jr.: Recent Advances in the Study of Sulphide Enrichment, *Mining and Scientific Press*, vol. cviii, No. 4, p. 172 (Jan. 24, 1914).

¹⁰ A. F. Rogers: Chalcocite Formation at Butte, Montana, *Economic Geology*, vol. viii, No. 8, p. 788 (Dec., 1913).

or 30 ft. south in the horse-tail veinlets and in the country rock. In this connection, it must be remembered that crystals and minor lenses of the cupric sulphide have been found in the horse-tail ore, hundreds of feet south from the north wall of the Leonard vein. The porosity of the enargite, slight fracturing of the veins in the country rock, and replacement of the quartz-monzonite and vein minerals, provided space for the crystallization of covellite.

Too much stress cannot be laid upon the penetrating power of mineralizers. When apparently solid country rock has been highly altered a hundred or more feet from the veins or fracture systems, we must conclude that a mineralizing solution or an igneo-aqueous vapor can, under the prevailing temperatures and pressures, penetrate porous veinlets of enargite and shattered areas. That local conditions in the mineralizers largely accounted for the formation of covellite, after the deposition of the enargite had ceased, may be seen in the fact that large amounts of enargite in the vein, above, below, and laterally, contain no covellite. Such conditions, together with structural and physical influences, have produced areas in the vein where other minerals have likewise been localized. Two hundred feet below this occurrence of covellite, persistent solid chalcocite veins, with only a slight sprinkling of pyrite, replace sericitized quartz-monzonite, at times, to a width of two or more feet.

Evidence of the replacement of enargite by solutions depositing covellite is found in the small and large remnants of enargite in covellite areas (see Fig. 8); in stringers of covellite through enargite crystals; and in irregular areas of covellite that have eaten into the interior of enargite masses. The action appears to have been mostly one of replacement, volume by volume. The enargite shows little evidence of fracturing.

Development almost wholly from enargite has left few characteristics of structure in the covellite. Even in apparently solid masses of crystal plates of covellite, unreplaced enargite may be seen with the naked eye upon close examination. Under the microscope, small and large portions of enargite are seen to be inclosed in the cupric sulphide. Covellite has at all times a very strong tendency to develop its own hexagonal or rhombohedral plates with perfect basal cleavage. These plates are found crossing the crystals of enargite in every direction and frequently grow to several inches in length, far exceeding the limits of enargite crystals, which seldom exhibit parallel orientation in adjacent individuals. That covellite has for the most part strongly preserved its own crystal form under conditions indicative of free crystallization, is everywhere apparent.

Shreds of the cupric sulphide are observable under the microscope and in large areas present peculiar phenomena. On etching with potassium cyanide, the structure of covellite is brought out, the cyanide salt eating into the cleavage cracks and leaving a rough dark network on the polished surface. The same salt attacks enargite in a similar manner. When areas

of covellite, that have resulted from a replacement of enargite, are etched, a series of lines parallel to the basal cleavage is developed. Minor lines in all directions complicate an indefinite rough etching between major directions. When enargite grains are included in the etched area, the orientation of the crystal systems may be compared. Enargite cleavage stands out prominently when etched with potassium cyanide. Crystals of the sulph-arsenate are commonly found in covellite areas with orthorhombic cleavage lines oriented in all conceivable directions to the basal etching of covellite. Moreover, one blade of cupric sulphide may cut across half a dozen or more crystallographic units of enargite, preserving its own structure throughout. That this mineral is strongly addicted to crystallize with its own form and structure is plain from the similarity of the covellite that has replaced enargite, pyrite, and country rock, to the covellite crystals formed in the free space of vugs.

In this connection the universal mottled structure of covellite is of interest. Observed variations in color, which may be due to several physical causes, are evidenced by irregular intergrowths and penetrations of a light and a deep blue. At times the variation appears to be the effect upon previously formed dark-blue covellite of later light-blue material. Again, change in color may be due to different orientation in the grains of cupric sulphide. The presence of grains of enargite not completely altered to covellite, but appearing light blue, gives rise to a mottled effect and on etching presents the typical enargite crystal structure. Later shady developments of bornite produce a mottled appearance in sections. Much of the variation in color, however, is believed to be due to the chemical constitution of the covellite.

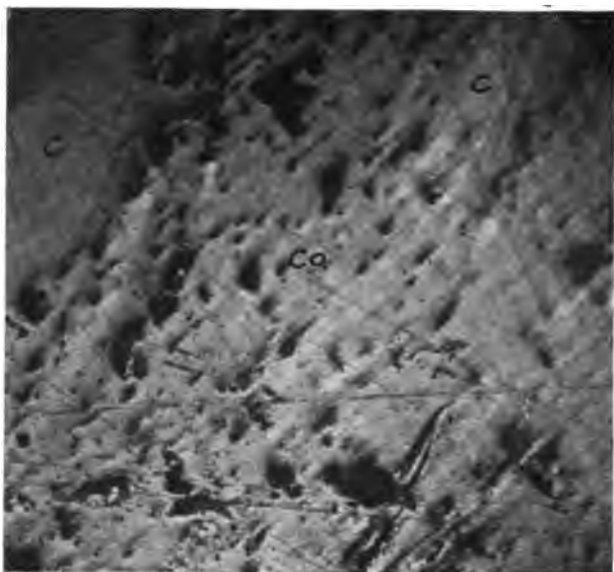
Sections of covellite blades typically show considerable finely divided gangue. (See Fig. 7.) To a large degree enargite presents the same phenomenon. (See Fig. 8.) The survival of the gangue in the covellite is probably due to the preservation, in the replacement of enargite by covellite, of the gangue particles already contained in the enargite. Some quartz may also have been deposited contemporaneously with the formation of covellite.

Subsequent to the deposition of the cupric sulphide, a deformation of slight extent ensued, which in places bent, buckled, and fractured the blades, forming, along and across cleavage directions, angular spaces and lenticular gashes, which have been filled with later primary bornite and chalcocite. (See Fig. 11.) These later primary sulphides, including minor amounts of chalcopyrite, have greatly altered the covellite lenses.

Bornite.—Following the cessation of covellite deposition, and the later deformation of the vein minerals, bornite was precipitated from solutions, filling interstices in and between covellite, enargite, pyrite, and quartz, and at the same time replacing these minerals, particularly the covellite. The condition of bornite in all the Butte ores is very unstable. It evidently

replaced the enargite and covellite, at times to a considerable extent, but, being unstable in the presence of later chalcocite mineralizers, has almost disappeared. Its relation to chalcocite is very intimate and often difficult to unravel.

During the process of replacement of enargite and covellite, the gangue underwent little alteration. Its presence, in irregular narrow lines of bornite in chalcocite areas, has served to identify bornite as an earlier mineral. When all of the previous copper sulphides are replaced by chalcocite, the gangue disappears as well. In this respect the solutions depositing glance are believed to have been more energetic reagents than the earlier mineralizers.



C, chalcocite; Co, covellite.
1,600-ft. level, Leonard mine, Leonard vein.

FIG. 11.—SHATTERING OF COVELLITE BLADES AND CEMENTATION AND REPLACEMENT BY CHALCOCITE. $\times 80$.

Coincident with the replacement of covellite, a great deal of bornite developed as an intermediate stage in the alteration of older minerals to chalcocite. Bornite appears in shadowy, irregular areas and has, to an advanced degree, been replaced by glance. Often, after etching polished slabs of bornite and chalcocite which have originated by replacement, the two minerals are seen to grade into each other, with very delicate changes in color, and no definite demarkation between one mineral and the other. The copper-iron sulphide is found mostly as a rim around plates and remnants of covellite, as irregular areas at the contact of covellite and glance, and peppered through glance in very small grains, remnants which sur-

vived the processes of replacement. This later occurrence gives a mottled appearance and a rough relief to the chalcocite.

In the case of sections where bornite appears to have been formed definitely earlier than chalcocite, it is believed that when the ores were being deposited, the mineralizers were at first constituted so that bornite began precipitating out as the covellite was replaced, by reason probably of the abundance of iron. As this saturation for bornite lessened or the iron contents diminished, chalcocite was precipitated. That such a condition prevailed is evidenced by the earlier masses of bornite, the crystallographic intergrowths of the two minerals in the Leonard ore, and the universal primary intergrowths. Considering the conceded primary nature of bornite, these developments are strong evidence for the primary nature of the chalcocite and covellite. As the determining iron factor in the solutions lessened, replacement of covellite by chalcocite took place directly. It appears, however, probable that, even during the last stages of chalcocite deposition, bornite continued to be formed in small quantities.

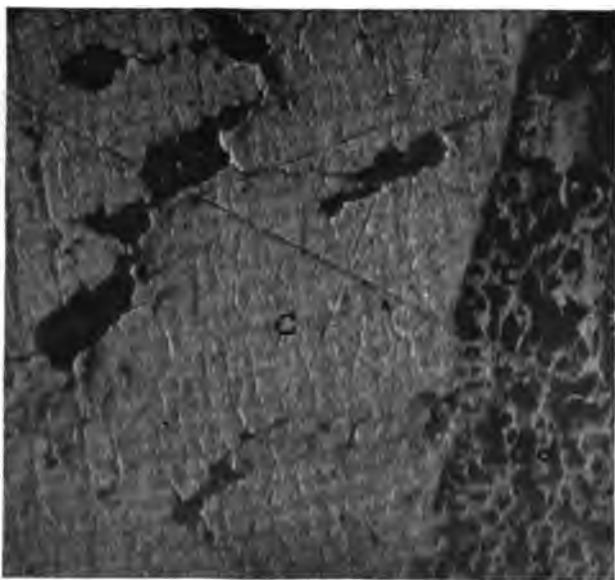
In this period of alteration occurred a very minor development of primary chalcopyrite, which is believed to have been intermediate in the alteration of covellite and pyrite by chalcocite mineralizers, thus occupying a position like that of bornite, which it preceded or followed. Most of the chalcopyrite in the sections is formed near pyrite grains. These pyrite grains are believed to have furnished some iron for the formation of chalcopyrite and bornite by the ascending primary solutions.

As is evident in other Butte veins, much bornite is deposited directly from primary solutions, as an enrichment and as a cementing material of quartz, pyrite, and enargite. When this is the case, chalcocite as an intergrowth or later primary development, is universally present. The common form of appearance of bornite is a result of deposition from primary solutions, while these mineralizers are altering the previously formed vein minerals and forming new ones. Under these conditions bornite occupies an unstable position, readily yielding to copper enrichment or impoverishment of iron contents, to form the final mineral of the reactions, chalcocite. In contradistinction to this process of enrichment, bornite frequently forms in the impoverishment of rich ores, notably covellite and chalcocite, as an intermediate product before the end reaction to chalcopyrite.

Chalcocite.—With the exception of some later chalcopyrite, chalcocite has been the last copper mineral to form in the Leonard vein, from ascending primary waters. Large and small stringers are seen running through solid, sericitized quartz-monzonite, as a direct replacement of the rock-forming minerals. Shattering of the country rock and vein minerals afforded channels for the entrance of mineralizers. Shots of the cuprous sulphide occur peppered through some areas of country rock. It is cer-

tain that these agencies depositing glance were very strong in their effect on the quartz-monzonite minerals and those composing the veins.

Chalcocite is found traversing all of the earlier sulphides in veinlets and in replacement areas. Contacts of the earlier minerals and chalcocite are always sharp and distinct. With the exception of occasional structure derived from covellite, the chalcocite on etching produces a fine example of primary cleavage. The replacement of enargite by solutions depositing chalcocite has been described under enargite. This action has been universal and is far advanced in places. Other veins of enargite are untouched by chalcocite. Local conditions in the mineralizers, due to



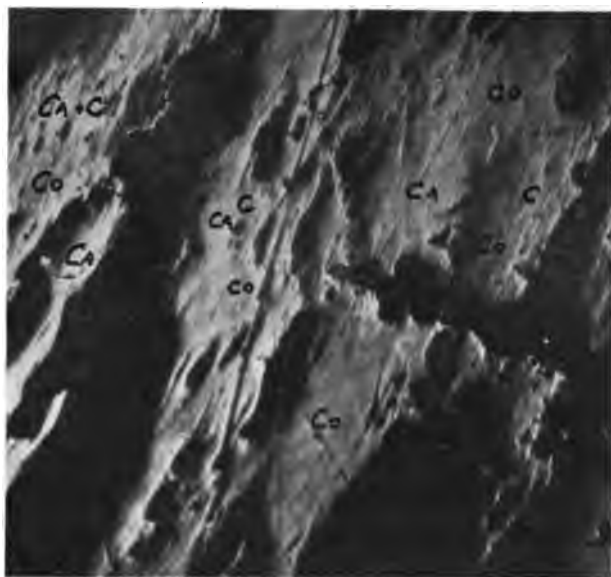
C, chalcocite; Co, covellite.
1,600-ft. level, Leonard mine, Leonard vein.

FIG. 12.—PRIMARY ETCHING STRUCTURE OF LEONARD CHALCOCITE REPLACING COVELLITE. $\times 80$.

concentration of solutions, chemical composition, temperature, pressure, and permeability of veins and wall rock, account for great developments of glance at some levels and minor amounts of it at no great distance in others. Bornite, covellite, enargite, and pyrite, represent the order of ease in which chalcocite replaces the vein minerals. Whole areas of bornite have been completely replaced. Advanced stages in the replacement of covellite prove that a complete transition took place from covellite to chalcocite. Similarly, enargite has been corroded until, as in the case of pyrite, rounded remnants alone attest the former crystals.

Fracturing after the deposition of covellite allowed access of solutions

period of mineralization. The primary solutions carrying the cuprous sulphide first deposited bornite, with iron as the determining factor, until, by the reduction of the bornite molecules, the chalcocite began to precipitate as the earlier minerals were replaced, forming crystallographic intergrowths and primary interlockings with bornite. Finally chalcocite itself replaced the previously formed vein minerals directly and filled cavities, bornite forming at the same time in small amounts. In most of the observations on the occurrence of chalcocite it has been found to be a replacement of quartz-monzonite and vein minerals. Instances of chalcocite and quartz filling of vugs have been observed in the Gagnon mine and chalcocite vugs are occasionally found.



C, chalcocite; Ca, chalcopyrite; Co, covellite.
Leonard mine, Leonard vein.

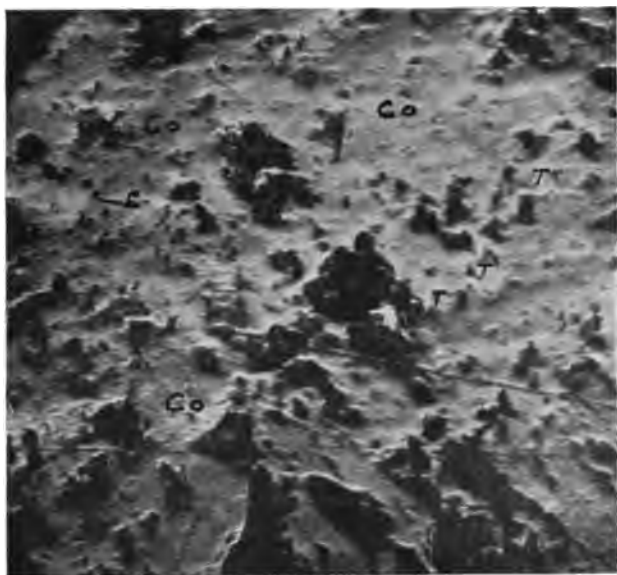
FIG. 14.—CHALCOPYRITE REPLACING COVELLITE AND CHALCOCITE. $\times 80$.

In the replacement of covellite, glance and bornite have played a very intimate part. Covellite has altered directly to both bornite and glance, bornite being the first mineral formed as a result of this reaction. Later solutions not only replaced covellite and deposited glance, but also attacked and replaced the previously formed bornite. It is believed also, that at the time mineralizers were replacing covellite and depositing chalcocite in its place, a universal but finely divided development of bornite occurred. Fig. 13 shows a typical case of almost complete replacement of former covellite masses by chalcocite. Remnants of covellite can still be seen.

believed to be the local result of the action of the later mineralizers on the iron sulphide. Chalcopyrite altering to chalcocite with an intermediate rim of bornite occurs on the 2,400-ft. level of the Anaconda mine. (See Fig. 15.) Linings of bornite surround some of the chalcopyrite of the Leonard vein. At other times the contacts of chalcopyrite veinlets are sharp and distinct.

3. *The Fault-Vein Covellite*

The Edith May vein in the Speculator mine, the Gray Rock vein in the East Gray Rock and Diamond mines, and the South Bell vein in the High Ore mine, have revealed covellite in scattered lenses from several



C, chalcocite; Co, covellite; T, tetrahedrite.
800-ft. level, Speculator mine, Edith May vein.

FIG. 16.—TETRAHEDRITE IN COVELLITE AND CHALCOCITE STRINGERS. $\times 80$.

inches to several feet in extent. All of these fault veins belong to the Blue vein system described by Sales.

The Speculator Ore.—Sections of the Edith May ore present a matrix of minerals very similar to those of the E-W. Leonard vein. Pyrite, enargite, tetrahedrite, covellite, bornite, and chalcocite make up the ore.

Pyrite is present in crystals and in partly corroded grains. Being the first sulphide to form, it has undergone continued attack by later solutions. Enargite is found in irregular remnants, almost wholly replaced by covellite, tetrahedrite, and chalcocite. Practically all of the covellite is

believed to be a replacement of enargite. Before the deposition of covellite, small but widespread quantities of tetrahedrite replaced enargite. Tetrahedrite is usually found in irregular small masses, not taking the form of veinlets. (See Fig. 16.) It is evidently a precipitation following enargite before the beginning of covellite deposition, although some areas have the appearance of intergrowths with covellite. Tetrahedrite strongly resists alteration.

Covellite contains many small particles of gangue. It is chiefly the result of the replacement of enargite, but also of crushed country rock and pyrite. The cupric sulphide is much mottled, partly by reason of a faint coloring of enargite that has almost completely altered to covellite. On etching with potassium cyanide, the enargite remnants are attacked more markedly than the covellite. In this manner some of the covellite is seen to retain the structure of the enargite it has replaced. A considerable proportion of the mottled appearance is due to the alternation of covellite with a slightly lighter colored mineral, probably a lighter colored phase of the same mineral, as both varieties etch similarly.

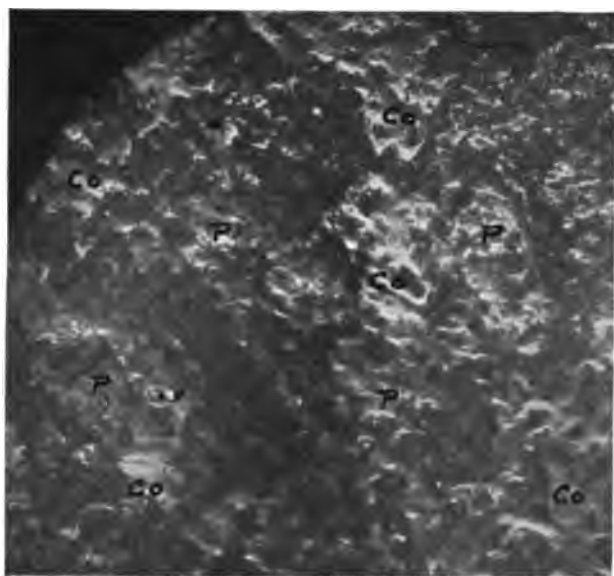
Bornite occurs occasionally in veinlets through covellite and as a fringe along the contacts of chalcocite and covellite. It is an intermediate product of the alteration and is sparingly developed. Chalcopyrite, with the bornite, is the outcome of a local concentration of iron in the mineralizers, due sometimes to the presence of pyrite in the ore. Bornite is also found as shadowy specks in areas of covellite.

In some sections the alteration of covellite has not begun. Others show that replacement by chalcocite has occurred to a slight extent. In all directions minute branching veinlets of glance, with sharply cut borders, are characteristic. Very little evidence remains of a shattering of the vein later than the deposition of covellite. In the replacement of the cupric sulphide by primary chalcocite, the sharp contacts of the veinlets, the crystalline etching structure of the chalcocite, and the absence of gradation of intermediate alteration products between the center of the veinlets and the covellite, signify its primary character. All of the minerals in this ore are undoubtedly primary. The development of tetrahedrite and a much smaller proportion of chalcocite, distinguish the ore from that of the E-W. Leonard vein.

The East Gray Rock Ore.—The specimens preserved of the ore minerals in the East Gray Rock vein show a uniform variety of constituents. The ore is a result of precipitation of minerals from solutions or igneo-aqueous vapors, as they ascended through the country rock between strong clay walls of the vein and replaced the crushed material. As in the Leonard vein, quartz and pyrite constituted the first vein filling. Characteristic of this approach to the border zone between the copper and zinc veins as established by Sales, different minerals begin to appear. Sphalerite ranks next to pyrite in the paragenesis of the vein minerals,

while covellite has universally replaced the gangue, pyrite, and zinc blende, at the same time precipitating directly from solution. Tetrahedrite is another mineral, uncommon in the Leonard ore, but abundant in the Gray Rock vein as an earlier mineral, an intergrowth with covellite and a later replacement of previously formed vein minerals. Minor amounts of chalcocite replace all other minerals, excepting tetrahedrite.

A feature prominent in the Gray Rock ore is the thorough shattering of pyrite and gangue before the deposition of covellite. Even minute pyrite grains show extensive shattering, the crystals being cemented and the cracks filled with covellite. In the course of enrichment, pyrite has been altered to several minerals, chief of which is covellite.

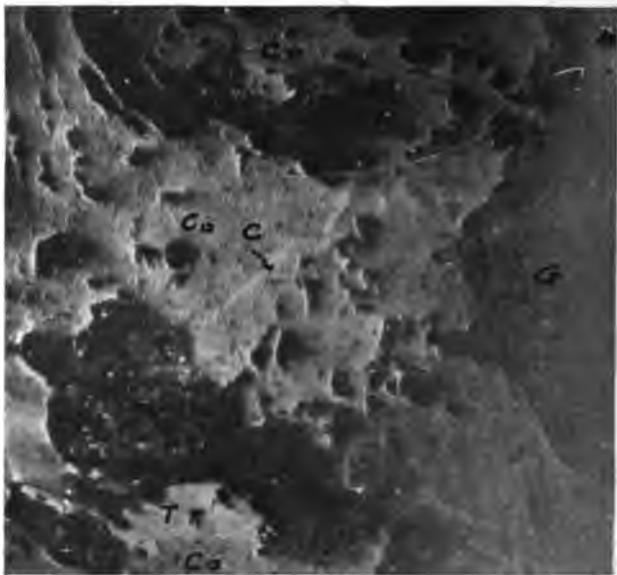


Co, covellite; P, pyrite.
East Gray Rock mine.

FIG. 17.—COVELLITE CEMENTING CRUSHED PYRITE GRAINS. $\times 80$.

Many micro-crystalline specks of pyrite are peppered through the covellite, most of them remnants of former larger grains. (See Fig. 17.) An unusually well developed mottled structure of the cupric sulphide, caused by intergrowths of a light and a dark blue material, is found in this ore. The lighter blue portion of the mineral forms usually around inclusions of gangue. Etching develops characteristic covellite cleavage and a difference in the crystallographic orientation of blades of the mineral. Covellite has resulted from the replacement of pyrite, sphalerite, and gangue, and from a direct crystallization from solutions.

Features contrasting with the Leonard ore are: the absence of enar-



C, chalcocite; Co, covellite; G, gangue; P, pyrite; T, tetrahedrite.
East Gray Rock mine.

FIG. 18.—TETRAHEDRITE INTERGROWTH WITH CHALCOCITE AND REPLACING COVELLITE. $\times 80$.



C, chalcocite; S, sphalerite.
East Gray Rock mine.

FIG. 19.—CHALCOCITE REPLACING SHATTERED SPHALERITE. $\times 80$.

gite; sprinklings of irregular small splotches of tetrahedrite; and little chalcocite. Tetrahedrite occurs as a deposition following sphalerite, as an intergrowth with covellite and, to a slight extent, as a replacement of it. It is generally intimately associated with gangue, as if clusters of quartz grains had favored its formation. (See Fig. 18.)

Chalcocite performs its usual function of replacing previously formed vein minerals. The apparent affinity of chalcocite for zinc blende has resulted in an advanced stage of replacement of the blende. (See Fig. 19.) Covellite also has totally replaced areas of sphalerite. The primary structure of the chalcocite, developed by etching, adds microscopic evidence to the structural proof of the primary nature of all of these sulphides.

In the tetrahedrite occur exceedingly fine specks of an unknown bright mineral, which familiar micro-chemical tests suggest to be an arsenical or antimonial sulphide of silver, replacing the tetrahedrite.

4. Other Occurrences of Covellite

Rare instances of the formation of small bunches of covellite crystals have been met in numerous mines. The Mountain View and Tramway have been interesting in this respect. Pyrite crystals in Tramway specimens have been determined to be later than the covellite, and to be a filling between the blades. From the mines of the central copper area, the covellite, in general, is found to be altered in varying degree to chalcocite and later chalcopyrite.

Stopes in the High Ore mine, on the South Bell vein, a member of the Blue vein system, sometimes open up small clusters of covellite blades. These masses of the cupric sulphide are found to be replaced by chalcocite to an advanced degree, without the development of bornite or chalcopyrite. Remnants of pyrite crystals speck the copper minerals. Bunches of this material have recently been found, 2,400 ft. below the surface, in the South Bell vein. Included chalcocite exhibits the typical etching of primary chalcocite. The covellite of the fault veins may, in general, be said to contain more impurities than that of the E-W. veins, probably on account of the difference in solubility of the materials replaced in its formation. Insoluble impurities may be almost wholly referred to the greater percentage of the rock-forming minerals only partly replaced in the formation of the fault-vein ore and the simultaneous deposition of silicates with the covellite.

Further from the central copper zone, even in the infrequent covellite specimens obtained, the presence of minerals typical of the intermediate copper zone is noted. Tetrahedrite and sphalerite are most frequently met. Covellite from the Bell mine, of the intermediate zone, contains small irregular masses of tetrahedrite, with branching veinlets, probably an

intergrowth with the cupric sulphide and in part a later deposition. The mottling of covellite in this part of the district is pronounced. The replacement of a previously formed pyrite vein filling is universal.

III. CONCLUSIONS

1. *The Replacement of Minerals*

From microscopic evidence it is clear that the primary minerals of the veins have been subjected to the action of solutions penetrating them under conditions rendering the minerals unstable. It is also evident that the mineralizers entering the veins carried with them different elements, some of which, under the imposed conditions, were stable after reacting with the vein materials. Exchange of constituents of the mineralizers with those of the previously formed vein minerals resulted in the deposition of minerals, stable under the prevailing conditions. Observed facts point to the interchange of chemical constituents of the minerals and later solutions and deposition of new minerals, occupying the same volume as the minerals replaced. The process of replacement appears to have progressed under conditions of simultaneous dissolution and deposition, with the formation, in some cases, of minerals intermediate in composition.

Local conditions of replacement, that may be present only in the contact film between the solutions and the minerals, are responsible for varieties in mineralogical developments. Variable activity in circulating mineralizers has influenced chemical composition and volume changes in the process.

Evidence of replacement with no change in volume is found in patches of chalcocite in large areas of covellite. Here the solutions have penetrated covellite, adding copper or subtracting sulphur necessary to form cuprous sulphide. Covellite areas in pyrite grains, exactly filling the dissolved areas, chalcocite in pyrite grains and gangue, chalcocite in enargite crystals, all attest to the replacement by volume, the carrying away of mineral substance and addition of new material. Such replacement is believed to be due to slowly moving and arrested mineralizers.

Irregular veinlets of the later sulphides in the earlier vein minerals chronicle replacement by volume. Further replacement has taken place around the edges of the earlier minerals, destroying surfaces and crystal-line outline. No space of solution ordinarily can be observed, at the contacts of replacements.

Replacement of bornite by chalcocite presents sub-microscopic phenomena, which, on account of the intimate relations of the two minerals, vary from the common conditions of the process. Etching of primary chalcocite and bornite, in sections where covellite has been replaced by

both, shows a good primary structure. The chalcocite etches with fairly uniform rectangular cleavage, the distinctive feature of its habit being the almost uniformly straight boundaries to the individual grains. Where bornite has been replaced the gradation of one mineral into another is sometimes very indistinct. The relations of these minerals are remarkable in this respect. Cleavage of chalcocite extends into grains of bornite, but is not so pronounced when pure bornite is encountered. A zone of incipient replacement develops good cleavage. It appears that the chemical interchange incident to the metasomatism of bornite, by solutions depositing chalcocite, bears a markedly close relation to the common molecular constituents of the two minerals and by the elimination of some molecules of Fe_2S_3 the percentage of chalcocite present is raised by successive gradation up to that of the solid chalcocite.

Such a type of structure is frequently difficult to interpret. Doubt may exist concerning the priority of minerals, but convincing proof is present in the fact that the metasomatism incident to the formation of bornite from covellite or enargite did not proceed to the replacement of the particles of gangue disseminated through the primary minerals. Thus the bornite contains remnants of gangue, while the action of the chalcocite mineralizers replacing bornite destroyed the included gangue. It is an observed fact, that in primary chalcocite a comparatively pure mineral may be expected in whatever quantity it may be present. Pyrite, the most common impurity in chalcocite, is more resistant to the action of the depositing solutions, but the presence of any quantity of gangue or ore minerals, excepting the intimately related bornite, is rare.

In replacement by primary ascending solutions different structures are involved from those of replacement by descending solutions. In the former case, the contacts of the minerals replaced and the minerals deposited are so sharp that microscopic examination reveals no solution line. If intermediate alteration products are formed, the same condition holds true. But in the case of alteration by descending waters, a hazy line of intermediate products with indefinite contacts, generally marks the zone of metasomatism. Such sharpness of outline in replacement by primary minerals is believed to be due to greater chemical activity of the solutions, a result of their chemical constitution, and the presence of high temperatures and pressures.

2. The Character of Mineralizing Solutions

Sales has outlined the probable condition of magmatic waters and igneo-aqueous vapors in their ascent through the fissures in the quartz-monzonite. He concludes that the earlier solutions were preponderantly alkaline, by reason, in some measure, of the subtraction of sodium, calcium, and manganese from the quartz-monzonite wall rock. Later solutions were less alkaline, since the altered wall rock and the deposition of

quartz acted as an insulation for the solutions; and accordingly the percentage of hydrogen sulphide in the mineralizers increased. As a result of this change the final mineralizing solutions depositing chalcocite might be relatively poor in alkaline salts. Whatever was the constitution of these later solutions depositing chalcocite, it is certain that they were more active than the preceding mineralizers in replacing the most difficultly altered minerals. Final solutions rising in some parts of the veins were at times highly alkaline. The deposition of calcite resulted occasionally and that of barite commonly.

Paragenesis and Character of the Vein Minerals

The entrance of early minerals proceeded along previous fractures and altered the quartz-monzonite, forming solution channels and vugs. The greatest alteration took place along the main zone of fracturing.

Quartz and pyrite were the first minerals to be deposited, mainly as a replacement of quartz-monzonite. Enargite followed, and all three minerals continued to form. Covellite, chalcopyrite, bornite, and chalcocite followed in the order given.

Ferro-magnesian minerals of the quartz-monzonite are more readily attacked and replaced by solutions of the sulphides than the feldspars or quartz.

The same order of deposition is found in both E-W. vein ore and fault-vein ore, with the addition of tetrahedrite in the fault veins, of the same age and sometimes just previous, and continuing till slightly after the crystallization of covellite.

Quartz and pyrite depositing first were shattered and cemented by later minerals. Enargite shows little shattering, covellite generally a slight buckling of plates.

Covellite is a replacement mainly of enargite, other vein-forming and rock-forming minerals, and frequently crystallizes in the free space of vugs. In general, it strongly preserves its own crystal form, but sometimes shows internal structure after enargite.

Irregularity of the occurrences of covellite is regarded as due to local conditions in chemical constitution of the mineralizers, local physical conditions, chemical character of inclosing medium, arrested movement of solutions, and variation in temperature and pressure.

Slight shattering after deposition of covellite afforded channels for entrance of solutions depositing glance and bornite.

Mineralizers depositing chalcocite were very active in the replacement of previously formed vein minerals and country rock, more so than previous or later solutions. Chalcocite belongs to one continuous period of primary deposition and on etching is generally found to preserve its own internal structure. Smaller amounts of it occur in the covellite of the intermediate zone.

Bornite and chalcopyrite were developed with the entrance, through minute fracture planes and cleavages in earlier minerals, of solutions carrying the cuprous sulphide. Local concentration of iron in the mineralizers, sometimes due to vein pyrite, caused deposition of these two minerals.

The relation of bornite and chalcocite is very intimate. In the case of the Leonard ore, bornite formed before the chalcocite and continued to deposit simultaneously till enrichment ceased.

Bornite is very unstable and easily attacked, forming chalcocite and chalcopyrite. Bornite, covellite, sphalerite, enargite, pyrite, and quartz, represent the order of ease in which chalcocite replaces the vein minerals.

Tetrahedrite is present in the covellite of the intermediate copper zone but not in the central copper zone material, evidence formerly presented by Sales in describing change in character of mineralization outward from the central copper area.

Solutions of later sulphides have generally attacked and replaced earlier-formed vein minerals and quartz-monzonite.

In the alteration of pyrite to chalcocite an orderly succession of copper addition and iron subtraction occurs, to form chalcopyrite, bornite, covellite, and chalcocite.

The order of resistance to the alteration of primary sulphides by later mineralizers is: pyrite, tetrahedrite, enargite, covellite, chalcopyrite, bornite.

Differences in the distribution of minerals in the veins were probably due to highly local variations in the constitution of the mineralizers. The physical and chemical effects of the wall rocks and previously formed vein minerals were practically the same through large areas of orebodies where variations in mineralization occur.

Secondary bornite and secondary chalcopyrite are plentiful as associates of secondary chalcocite. They are frequently preserved as a halo around pyrite grains or residual as intermediate products in the descending secondary sulphide enrichment of pyrite.

The primary character of chalcocite is recognized under the microscope by eutectic patterns with recognized primary minerals, by primary intergrowths with the same minerals, by physical relations with primary minerals and by distinctive structure developed by etching.

The secondary nature of chalcocite is determined by decomposition cracks and characteristic etching structure.

The author is indebted to Reno H. Sales, Chief Geologist of the Anaconda Copper Co., for advice and criticism in the preparation of this paper and to Prof. D. C. Bard, of the Montana State School of Mines, for rare specimens and the loan of apparatus.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Duplex Process of Steel Manufacture at the Maryland Steel Works

BY F. F. LINES,* SPARROWS POINT, MD.

(San Francisco Meeting, September, 1915)

It is not the intention of the writer to enter into a discussion of the relative merits of the duplex process as compared with the straight scrap and pig iron process, working under the same conditions, but rather to present in a general way several modifications of the former process, and then describe the method adopted by the Maryland Steel Co., as best suited to local conditions, and also to meet the metallurgical problem presented in the removal from the iron of an unusual constituent—chromium.

The principal factors to be considered in deciding whether or not duplexing should be employed are:

- (1) Chemical constituents of pig iron.
- (2) Variety and quantity of steel to be produced.
- (3) Relative cost of pig iron and scrap.
- (4) Regularity of operation.
- (5) Capital to be invested.
- (6) An existing Bessemer plant.

Having decided to adopt the duplex process, several combinations may be used, viz.:

- (1) Acid-lined converter with basic open hearth.
- (2) Acid-lined converter with electric furnace.
- (3) Basic-lined converter with basic open hearth.
- (4) Basic-lined converter with electric furnace.
- (5) Basic open hearth with electric furnace.

The combination of an acid-lined converter with a basic open-hearth furnace is the one most commonly employed, and as it is the one best adapted to the conditions encountered at the plant of the Maryland Steel Co., we will briefly outline the advantages and disadvantages of the different methods of working.

There are two general methods of blowing, as follows:

- (1) The charge is desiliconized and partly decarbonized in the con-

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verter. On account of the difficulty in regulating the percentage of carbon in the blown metal, this method is not desirable where the amount of phosphorus or other impurities to be removed is so low that the duration of the subsequent open-hearth operation is determined by the time required to adjust the carbon to the specification limits. This objection, however, does not hold when the blown metal is to be added to a part charge of pig and scrap previously melted in the open hearth, or when it is to be treated by the continuous process in furnaces of 200 to 300 tons capacity.

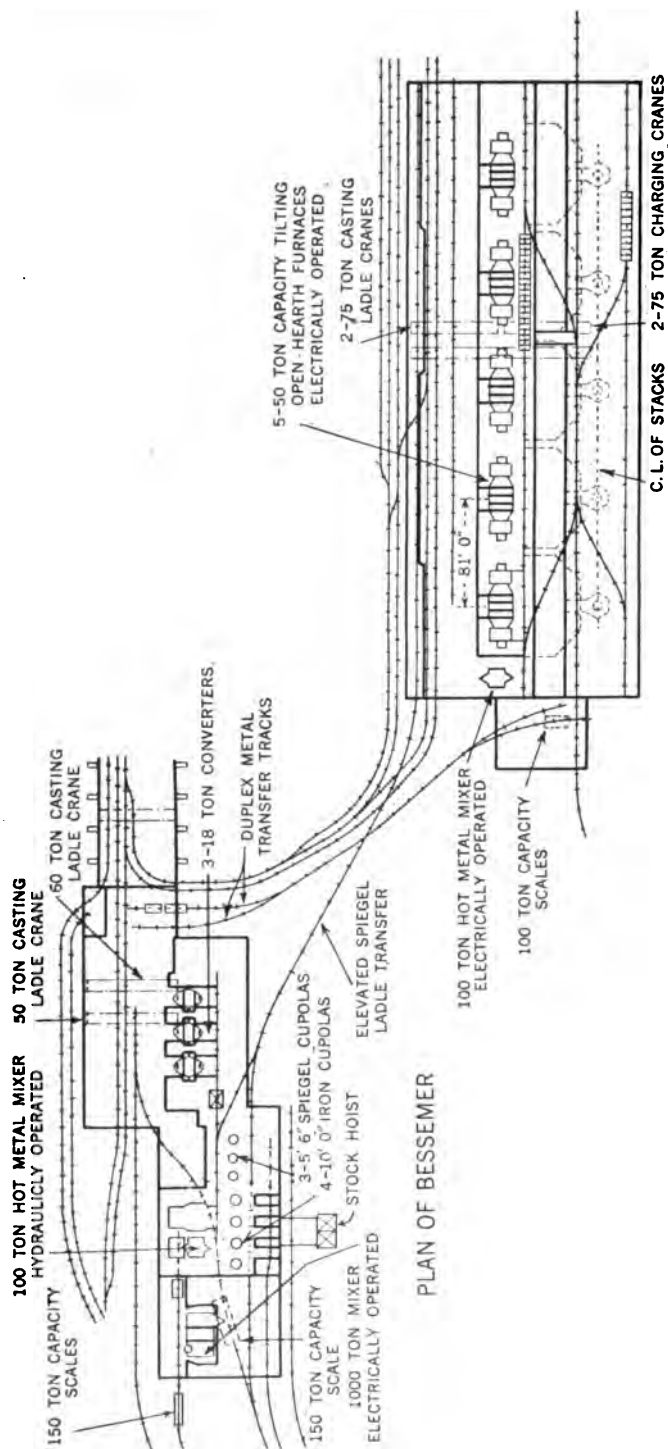
(2) The converter metal is completely desiliconized and the carbon blown down to about 0.10. This method of blowing commends itself on account of the regularity of the carbon content of the metal going into the open-hearth furnace, and in certain cases the more complete removal of other constituents.

Two methods of charging the open-hearth furnaces have been employed, dependent upon the facilities for handling the blown metal. In the first plants built it was necessary to send individual Bessemer blows to the open hearth. However, in the plants built recently it has been found advisable to assemble the blows necessary for an entire open-hearth heat in one ladle in the converting department. This method requires larger crane capacity, but has the great advantage of reducing the charging of the furnace to one operation. Both of these methods allow the slag to be kept out of the furnace. In deciding upon the most desirable size of an open-hearth furnace, the question immediately arises as to the type—tilting or stationary; and as this discussion is limited to duplexing, where 8 to 12 heats are tapped in 24 hr., there is little to be said in favor of the stationary furnace. Of late the tendency has been toward the installation of large tilting furnaces of 200 to 300 tons capacity, and while much of the evidence is contradictory, it can be safely stated that these larger furnaces show a saving in the cost of operation and repairs over the smaller ones. We are indebted to Dr. Schuster for a valuable paper read at the May, 1914, meeting of the British Iron and Steel Institute, in which he published the results obtained when working with different sizes and types of furnaces under identical conditions. As a result of experiments, covering a working period of 12 months, he has been able to establish conclusively that:

(1) The quality of steel produced is both physically and chemically independent of the type of furnace employed.

(2) The yield is practically the same whenever the same conditions are observed in furnaces of different types.

(3) Although the actual first cost of the 200-ton furnace is higher than the cost of the tilting furnace of smaller capacity, or of a stationary open-hearth furnace, this cost, calculated on the daily tonnage produced, works out most favorably for the 200-ton furnace.



PLAN OF OPEN HEARTH FURNACES
 Fig. 1.—GENERAL ARRANGEMENT OF CONVERTER AND OPEN-HEARTH DEPARTMENTS OF MARYLAND STEEL CO., SPARROWS POINT, MD.

(4) The fuel consumption is lower in the 200-ton furnace than in the other furnaces.

(5) The life of the refractory lining is longer in the 200-ton furnace.

(6) The labor of operating is less in the 200-ton furnace.

At the plant to be described, the following points decided the adoption of the duplex process: This plant was originally built to manufacture Bessemer steel rails, the blast furnaces and steel department being proportioned accordingly; the soaking pits in the rail mill were designed to treat the steady stream of hot ingots produced by the Bessemer process. To be able to supply open-hearth steel in place of Bessemer steel, the duplex process was adopted, using five 50-ton tilting furnaces, emptied at each heat. It was also decided at the start to blow the iron to about 0.10 carbon and to assemble the converter charges required for an open-hearth heat in a single ladle in the converting department.

The advantages of this plan for this plant are:

(1) Low first cost and no changes necessary in the existing plant.

(2) The ability to produce all Bessemer steel or all open-hearth steel, with very little change in the volume of finished product.

(3) The blast furnaces may be run to the best advantage, as no limit is set in regard to the silicon of the metal delivered to the converters.

(4) The change to duplex or to straight Bessemer steel is made without expense or loss of time.

(5) When working on open-hearth steel, the Bessemer department is fully employed.

The converting plant, built in 1889, the general arrangement of which with respect to the open hearth is shown in Fig. 1, is contained in one building. At the south end of the building is a 1,000-ton, cylindrical, electrically operated metal mixer. The iron from the blast furnaces comes to the mixer building in 45-ton ladles. A crane of 75 tons capacity, with an auxiliary 10-ton hoist for tipping the ladles, is used for handling the iron. The iron ladles coming from the blast furnaces are weighed on 150-ton standard-gauge track scales. Coke-oven gas or fuel oil is used for heating the mixer, a burner being arranged at each end for this purpose. However, it has been found necessary to use only one burner and the mixer has been kept in excellent condition by the use of 100,000 cu. ft. of gas in 24 hr. The mixer has a 9-in. magnesite lining, backed up by 13 in. of fire-clay brick.

Directly under the pouring spout is another standard-gauge track scale for weighing the metal charges for the converter. From the mixer, the molten metal is taken to the casting side of the converters. On its way it passes under the iron cupola runners, where it may be stopped to receive an addition of cupola metal. The converters are served by two traveling cranes, one with a 50-ton ladle hoist, having a 5-ton auxiliary

hoist for pouring the iron into the converters, and the other, a 75-ton crane which takes the blown metal from the converters.

The method of pouring the iron into the converters is shown in Fig. 2. The three converters are of the concentric type, of 18 tons rated capacity, but 20 to 22 tons of metal have been repeatedly blown. The converters, operated by hydraulic cylinders, are mounted on structural-steel columns and are 33-ft. centers. The converter bottoms have 24 tuyères, each tuyère having eight $\frac{9}{16}$ -in. holes, giving a total effective area of 47 sq. in. per bottom. With 21 tons of metal, the depth of the bath is 24 in. The blown metal from these three converters is received in a 60-ton transfer ladle, which is conveyed by a motor car to the open hearth, a distance of 350 ft. The ladle is carried to the proper furnace by one of the 75-ton charging cranes, and placed in position as shown in Fig. 3. This type of transfer ladle differs from those in general use for this purpose, in that the metal is not poured over the top of the ladle into the furnace, but is tapped through a 3-in. nozzle placed on the side.

The open-hearth building is 459 ft. long, and 138 ft. wide, there being a 65-ft. clearance on the pit side of the furnaces and 65 ft. on the charging side. On the latter side there are two 75-ton electric traveling ladle cranes for handling hot metal and duplex metal on the charging floor, and a 10-ton charging machine. A 120-ton metal mixer is placed in line with the furnaces, and is so arranged that the metal can be drawn out on the charging side as a "fill out" for dead soft duplex metal, or on the pit side for recarburizing the heats in the casting ladle. Scales are located on both sides of this mixer. On the tapping, or pit side, 75-ton ladles, handled by two 75-ton traveling cranes, are provided. The ladle cranes are supplied with two auxiliary hoists for handling the recarburizing metal. The pit slag is handled in steam-dumping ladles.

The gas-producer building runs parallel to the open-hearth building, and contains nine self-cleaning Hughes producers, as well as two dolomite kilns, and one crusher. The stockyard is located between the gas producers and the open-hearth building. The cold stock is loaded on a stock platform, adjacent to the charging floor, by a 10-ton magnet crane, which fills the boxes on the ground level from standard-gauge cars.

The dimensions of the furnaces are:

Length between chill plates.....	31 ft. 2 in.
Length on the metal line.....	24 ft. 0 in.
Width on the metal line.....	11 ft. 6 in.

The actual area on the metal line is 175 sq. ft., when made up to hold 50 tons. This area generally averages 220 sq. ft. and with this 60-ton heats are tapped. The regenerators are:

Gas.....	8 ft. 3 in. by 22 ft. by 10 ft. 4 in., or 1,815 cu. ft.
Air.....	13 ft. 3 in. by 22 ft. by 10 ft. 4 in., or 2,915 cu. ft.

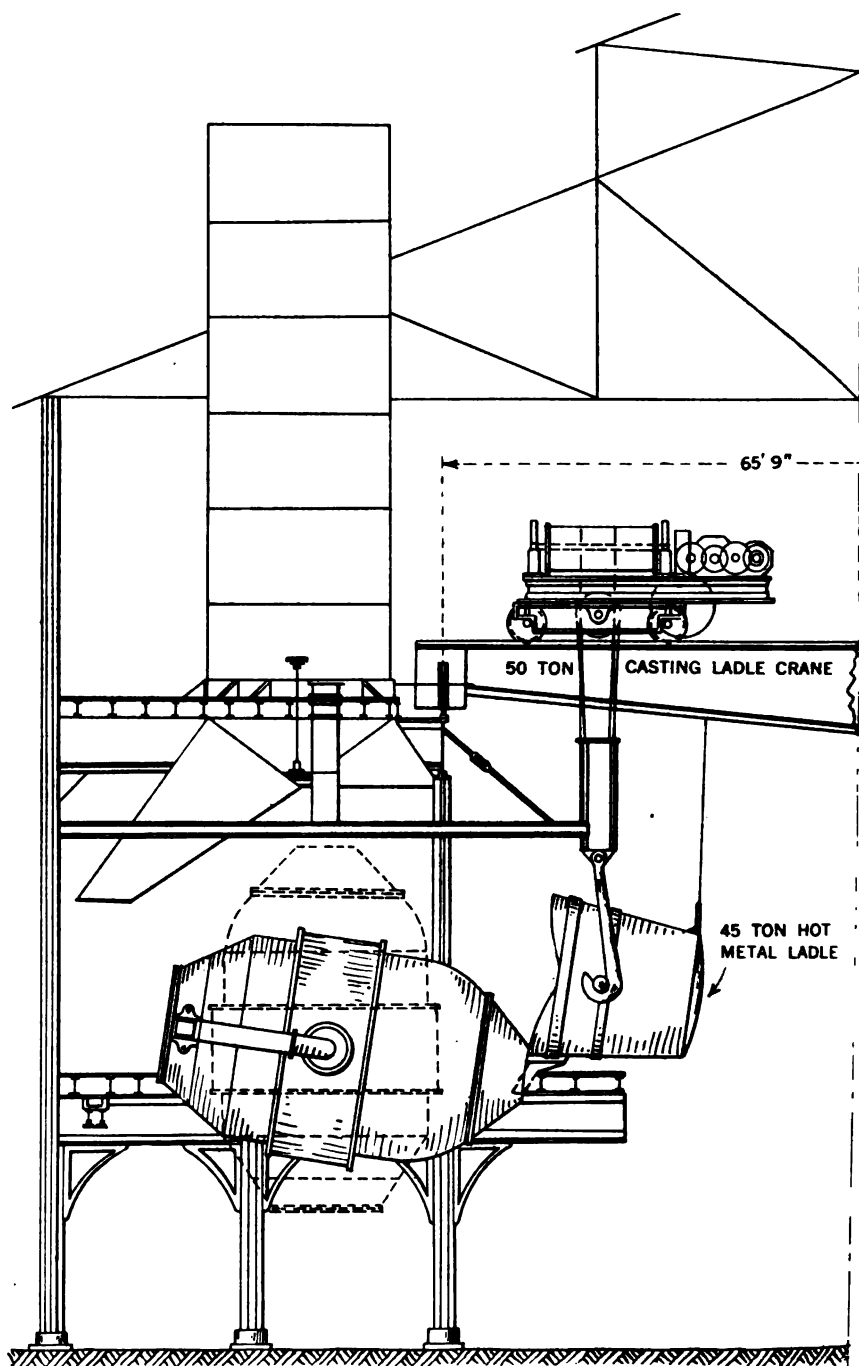


FIG. 2.—PART SECTION OF FIG. 1, SHOWING METHOD OF CHARGING THE CONVERTERS.

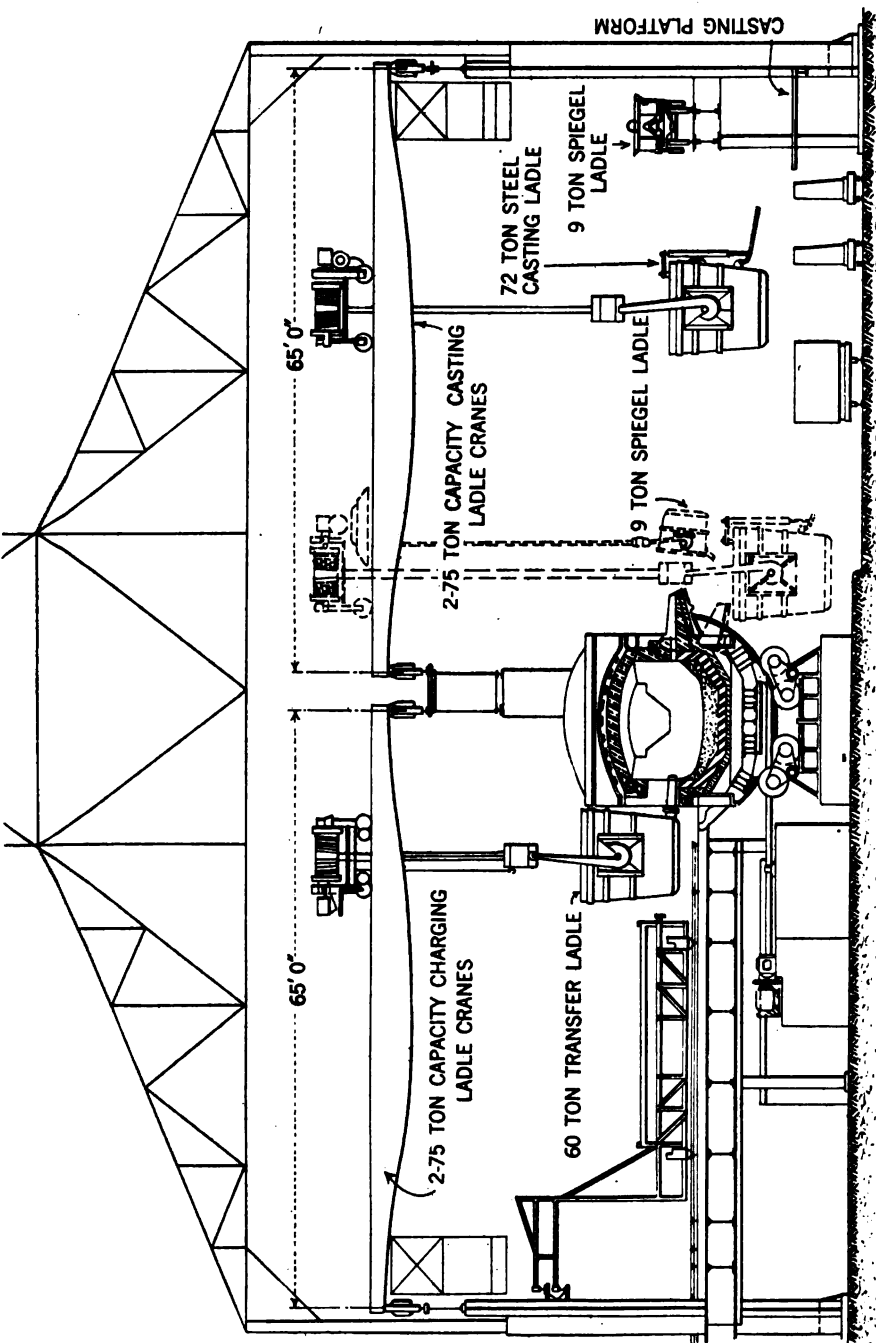


FIG. 3.—SECTION OF THE OPEN-HEARTH PLANT.

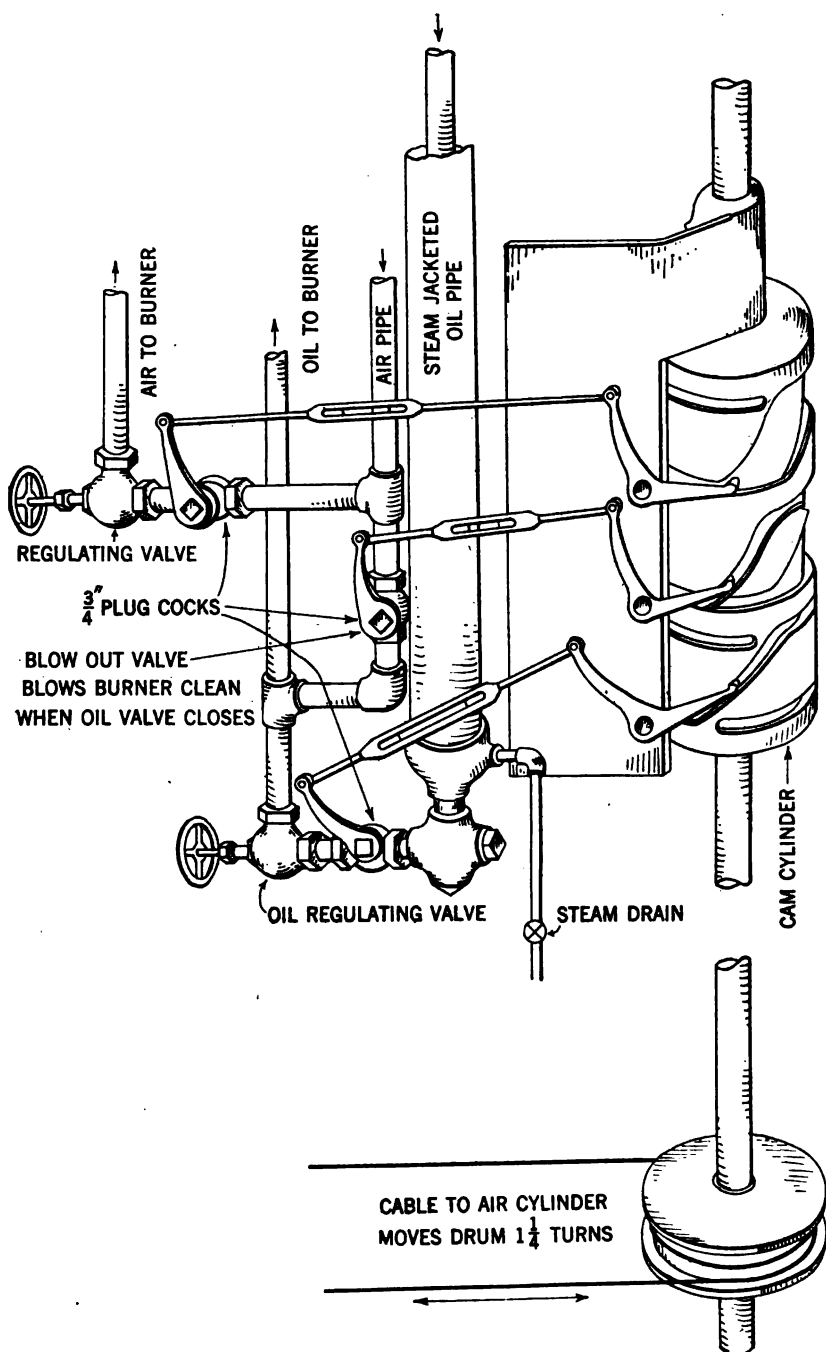


FIG. 5.—MECHANISM FOR CONTROLLING OPERATION OF OIL BURNERS.

for gas. This did away with the necessity of carrying the gas producers over the week end, with the attendant expense. It was also found that the furnaces could be brought to the proper temperature with oil more quickly than with gas.

Fig. 4 shows a furnace remodeled to burn oil. It will be noted that there are no reversing valves, these being replaced by an ordinary damper and air lid on each end of the furnace. The waste gases pass out through one downtake to the regenerator and through one flue to the stack.

Fig. 5 shows in detail the special mechanism by which with one movement at the reversal, the burner on one end is withdrawn from the furnace and the oil shut off, and at the same time the burner on the opposite end enters and the oil is turned on.

General Operation

For one duplex heat or one complete open-hearth furnace charge, three fully blown converter charges are poured into one transfer ladle, with or without the addition of iron in the ladle for carbon. In case the iron is not added in the ladle, the necessary carbon for proper action is added in the open-hearth furnace. By this method of assembling the open-hearth charge of blown metal into one ladle in the Bessemer, the complete charging time from the Bessemer to the open-hearth furnace is brought down to about 25 min.

Converter Charge and Scrap.—A typical converter charge is made up of 75 per cent. of mixer metal and 25 per cent. of cupola metal, the latter ordinarily being a mixture of 70 per cent. of pig iron and 30 per cent. of rough steel scrap. The direct iron, containing silicon, 0.40 to 1.50 per cent.; manganese, 0.70 to 0.90; phosphorus, 0.055 to 0.065; chromium, 1.75 to 2.50; and sulphur, 0.01 to 0.10 per cent., is poured from the mixer into 45-ton ladles, and the cupola metal is added in the same ladle. This ladle makes two trips to charge the three converters. On the first trip two converters are charged, both of which start blowing at once. On the second trip, the third converter is charged. By handling the iron in this manner, a minimum skulling of the ladle is obtained. The temperature during the blow is controlled by charging from 4 to 8 per cent. of bloom crops or rail ends through scrap chutes into the converters. This, with the scrap in the cupola metal, gives a total of 14 per cent. of scrap in the charge. The high percentage of scrap added to the charge is allowed on account of the heat developed by the chromium in its oxidation during the blow. From our practical experience, it is estimated that its heat of oxidation is one-half of that of silicon.

Method of Charging the Open-Hearth Furnaces.—As soon as the converter charges are blown they are poured into the transfer ladle, which is placed on a truck and taken to the open hearth, where it is weighed,

hoisted by the charging crane, and carried to the front of the proper furnace. No runner is used in charging the metal into the furnace, each transfer ladle being fitted with an 18-in. spout to direct the stream of metal from the nozzle. The ladle is placed within 2 ft. of the furnace door and the nozzle plate is knocked off. The helper, standing on a small platform on a level with the nozzle, picks out the small clay plug with a short hook, loosens the sand in the nozzle, with which the nozzle is closed, and the stream of metal spurts out at right angles to the ladle through the furnace door. As soon as all the blown metal has run out of the nozzle, and the slag appears, the ladle is tilted back and taken to the transfer ruck, where the slag contents are dumped into standard-gauge steam-tumping slag bowls, and the ladle is light-weighted. The slag in the bowl

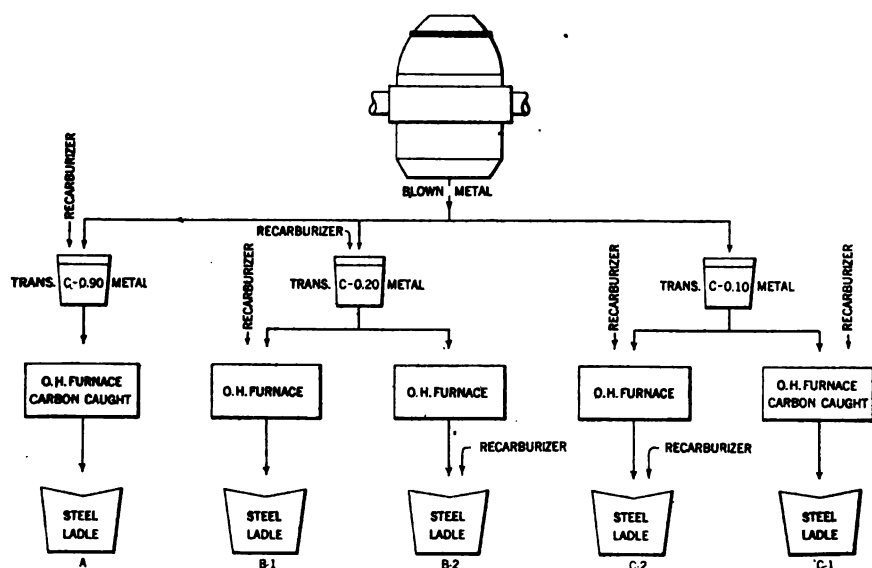


FIG. 6.—CHART SHOWING DIFFERENT POSSIBLE METHODS OF DUPLEXING.

is also weighed, and in this way the weight of the metal going into the furnace is established.

The time for running a heat from the ladle varies with the size of the nozzle, depending upon how many heats have been made through the same nozzle. It is the usual practice to take from six to eight heats off each nozzle. The total time from the pouring of the metal out of the converter nose to transferring all to the open-hearth furnace is from 22 to 28 min. There are few cases in which trouble is encountered in opening the nozzle on the transfer ladle. If trouble does occur, the metal is poured over the top of the ladle, or in case of steel in the nozzle, it is opened with a stream of oxygen. When pouring over the top of the ladle considerable acid-converter slag is carried into the furnace, which

interferes with the removal of the phosphorus and requires large lime additions for neutralizing.

Before charging, a mixture of 1 to 2 per cent. of burnt lime and $1\frac{1}{4}$ per cent. of roll scale is put on the bottom. It is desirable that the lime and scale have sufficient time to become thoroughly cut up before the duplex metal goes into the furnace. The time elapsing between tapping of one heat and charging the next varies from 20 to 60 min., an average of about 30 min. As soon as the duplex metal is in the furnace, a test is taken and sent to the laboratory, where the carbon is determined by direct combustion. The time elapsing between taking the test and obtaining the results from the laboratory is from 20 to 25 min. The treatment in the open hearth depends upon the carbon in the metal from the converters, as explained in the following description of the individual methods, outlined in the accompanying chart (Fig. 6).

Treatment in the Open Hearth

Method A.—The iron added for recarburizing the blown metal is taken direct from the mixer, and is about 20 per cent. of the total in the ladle. This method requires, with the particular iron dealt with, that the blown metal must not finish at too high a temperature, nor must the carbon be blown below 0.08, otherwise, in addition to an excessive blowing loss, there will result a violent ebullition in the transfer ladle when the recarburizing iron is added. This results in a mechanical loss due to boiling over the top of the ladle, and a loss in skulling of the ladle due to a lowering of the temperature of the steel. An advantage claimed for recarburizing in the ladle is that the addition of iron to the ladle reduces some iron from the slag back into the metal. However, a number of tests have been made on this point, and they do not show any appreciable reduction of FeO in the converter slag.

	Slag from Converter, Per Cent.	Slag in Transfer Ladle after Iron Added, Per Cent.
SiO ₂	22.80	24.60
FeO.....	24.16	22.60
MnO.....	7.70	7.00
Cr ₂ O ₃	27.65	26.10

We now have in the converter or transfer ladle 60 tons of metal of the following composition:

	Per Cent.
Carbon.....	0.70 to 0.90
Manganese.....	0.10
Phosphorus.....	0.065
Sulphur.....	0.040
Chromium.....	0.30 to 0.35

This metal as soon as it goes into the open-hearth furnace begins to take action. The metal loses about 10 points carbon and the phosphorus is down to 0.008 to 0.02 in the first test; the heat is, chemically, ready to tap. In the meantime, if the temperature of the bath is sufficient to go out on the first test, the melter has the tap hole open, the ladle hung, and if the analysis of the preliminary is within the limits the heat is tapped. In case the bath is not of the proper temperature, or the carbon too high, another test is already being worked by the laboratory, as the helper takes a test out of the furnace every 20 min. On receipt of the second result, if within the limits, the heat is tapped. The manganese, with this method, is generally added in the ladle, as well as the 50 per cent. ferro-silicon. If 11 per cent. ferro-silicon is used, it is added in the furnace 5 min. before tapping. A great many heats are of sufficient temperature to carry cold spiegel, added in the furnace 5 min. before tapping. Forty-five minutes is a good average time from charging metal into the furnace to tapping, and there is no doubt that this method has the advantage of rapidity and ease of operation, there being no further metal additions to the furnace, with their attendant delays and boiling effects in the furnace. The disadvantage of this method is that the time necessary to determine the carbon in the sample is of such duration that by the time the results are known the metal in the furnace has not the composition it had when the test was taken, and as the carbon does not drop uniformly, heats are tapped from time to time which do not come within the prescribed carbon limits for rail steel. This is of particular importance to this plant, as we have no other product into which heats unsuitable for rails can be rolled.

Methods B-1 and B-2.—These in practice were found to give unsatisfactory results, due to the metal skulling in the transfer ladle, and the fact that the small percentage of carbon often boiled out in the open-hearth furnace, before it was ready to tap, necessitating further recarburizing.

Method C-1.—The method of recarburizing in the open-hearth furnace up to 0.70 to 0.90 carbon and catching the heats "coming down" has no advantage, except that the handling capacity of the transfer ladles is increased by allowing them to be completely filled with blown metal, the recarburizing pig iron being subsequently added to the charge in the furnace.

Method C-2.—This method has been found to give more uniform carbon results in the finished steel. The iron is blown as previously described, but there is not the necessity of keeping the temperature within such close limits as when recarburizing in the converter ladle. The heats can be blown "younger"; the blown metal going into the furnace has the following composition:

	Per Cent.
Carbon.....	0.10 to 0.15
Manganese.....	0.10
Phosphorus.....	0.06
Sulphur.....	0.04
Chromium.....	0.20 to 0.30

The furnace, as in the other methods, has been charged with 2 to 3 per cent. of burnt lime and $1\frac{1}{4}$ per cent. of roll scale. This requires 30 min. before the duplex metal is charged. As soon as the blown metal is in the furnace, about 10 per cent. of the molten pig is added, which causes the previous dead soft metal in the furnace to take action. As soon as this iron is through the bath a test is taken, which should show approximately:

	Per Cent.
Carbon.....	0.15 to 0.30
Manganese.....	0.08
Phosphorus.....	0.008
Sulphur.....	0.05
Chromium.....	0.20 to 0.30

If the heat is hot enough and of the proper analysis, molten spiegel containing 20 per cent. of manganese, from the Bessemer cupolas, with enough molten iron from the open-hearth mixer to give the correct manganese-carbon addition to the heat, is poured into the casting ladle as the heat is tapped from the furnace, as shown in Fig. 3. A good average time for such heats through the furnace is $1\frac{1}{2}$ hr. This, with 40 min. between heats, gives 10 heats of 62 tons, or 620 tons per furnace, in 24 hr. In following this method, the metal in the furnace must be at a higher temperature than required for the first method, on account of the cooling effect of the addition of the recarburizer in the casting ladle. This requires the furnace to be carried at a higher temperature, with a corresponding shortening of its life. The percentage of lime added is necessarily higher on account of the necessity of carrying a more basic slag to counteract the addition of the recarburizer iron in the casting ladle, with its strong tendency to reduce the phosphorus in the slag back into the metal. A typical slag contains: SiO_2 , 14 per cent.; FeO , 16; MnO , 4; Cr_2O_3 , 7; CaO , 45; and P_2O_5 , 0.75 per cent.

Output

The maximum week's production at this plant averaged 600 tons per furnace in 24 hr. The maximum daily output per furnace has been 694 tons.

In case it should be necessary to duplex on all five furnaces to meet the increased blast-furnace output, allowed by the conditions of the market demand for steel products, the limiting factor in this plant would be the capacity of two pit cranes to handle the steel. In the final analysis of

any method or furnace, the correct factor of efficiency or speed is the tons of steel made per hour per square foot of hearth area of the furnace. From weekly averages made at different times, we find this to be 0.11 ton per hour per square foot of hearth area.

Fuel Consumption

When the furnaces were burning gas, no actual figures on coal per ton of ingots on duplex metal were available, but we are able from different runs to give some figures very close to the actual. The figures given for oil are actual meter readings.

	Duplexing, Including Heating Up	Cold Charge Including Heating Up
1913		
Coal per ton of ingots, pounds.....	235	1,030
Oil per ton of ingots, gallons.....	16	40
1914		
Coal per ton of ingots,	No coal used.	
Oil per ton of ingots, gallons.....	12.8	50

The results in 1914 were obtained under very unsatisfactory conditions, the plant operating only five days per week. Deducting the oil used in heating up the furnaces and considering the actual time the furnaces were making steel, we have an average of 9.3 gal. of oil per ton of ingots. Stated in terms of heat units per ton of ingots, we have 9.3 gal. of oil at 150,000 B.t.u. per gal., or 1,395,000 B.t.u.; assuming the available heat in the gas per pound of coal to be 10,000 B.t.u., we have $\frac{1,395,000}{10,000}$, or 139.5 lb. of coal per ton of ingots.

Life of Furnaces

On account of not duplexing exclusively on any of the furnaces, the following average for three years' operation includes about 33 per cent. of the heats made by the pig iron and scrap method: Furnace roofs, 300 heats; furnace fronts, 97 heats.

The average working time of three furnaces during this period was $5\frac{1}{2}$ days per week. During this time the plant had three extended shut-downs. From observations made as to the wear of the furnaces when duplexing for different periods, we are safe in stating that the furnaces duplexing exclusively will give 500 to 600 heats to a roof, and from 150 to 200 heats to a front.

Losses

In making comparisons between the losses in the duplex process and the cold-charge process, the fact must be remembered that the difference is due to the elimination of the carbon, silicon, etc., in the converter,

whereas the scrap in the ordinary open-hearth process has already withstood such a loss in its original conversion.

To satisfy the slag in Bessemer conversion 1.75 per cent. of the metallic iron is oxidized in the converter. This and the small loss in "shot" are the only losses sustained in the duplex process not common to both processes working with the same materials. The average loss from the Bessemer converter charge to open-hearth ingots for 1914 was 12.57 per cent.

With the introduction of the duplex process, there arose some criticism as to the quality of the resultant steel. As rail manufactures, we have had a most favorable opportunity of watching the behavior in track of 250,000 tons of duplex steel and 125,000 tons of ordinary open hearth. The results of observations of these rails, of practically the same chemical composition and made under uniform conditions of rolling-mill practice, warrant the statement that there is no difference in the quality of the two products.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Gasoline from "Synthetic" Crude Oil*

BY WALTER O. SNELLING, PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

IN the course of some experiments more than five years ago, made for a totally different purpose than the investigation of the oil used, I placed a small quantity of a transparent yellow lubricating oil in a bomb-like vessel and heated it to a relatively high temperature. At the end of the experiment I removed the oil from the vessel and was amazed to find that instead of bearing any resemblance to the oil which I had put in, it now had the appearance of ordinary crude oil. The green color by reflected light and the rich red-brown by transmitted light were so unmistakable as to at once lead to further investigation. I subjected the material to fractional distillation, and the surprise which I experienced at the *appearance* of the oil, changed to amazement when I found that it yielded, on distillation, 15 per cent. of gasoline and 30 per cent. of burning oil, and that its constitution resembled crude oil quite as much as did its appearance. Furthermore, the gasoline and kerosene distillates which it yielded were of a clear water-white color, entirely without treatment with acid or alkali, and were entirely free from the odor familiar in "cracked" petroleum distillates.

The result of this experiment was quite too remarkable to be credited without further confirmation, and I at once filled the vessel with some of the same oil that I had used before, and again heated to about the same temperature that I had previously used, and for the same period of time. Upon opening the vessel and removing the contents I found, not the material resembling crude oil that I had obtained before, but apparently only the same oil that I had put in, somewhat darkened in color, but nevertheless far different in appearance from the material obtained in the previous experiment.

Evidently some condition existed in the first experiment that had not

*This paper was presented informally and discussed at the New York meeting, February, 1915. Since the manuscript was received too late to print before the meeting, the paper will be presented at the San Francisco meeting, thereby giving an opportunity for further discussion.

existed in the second test, and here began a series of tests in which I sought by the change of one variable after another to arrive at the identical conditions which must have existed in the first experiment. Only the fact that the bottle of heavy oil used in the first test was still in its place, and the further fact that I had no crude oil among the materials at hand when I began the experiment—only these facts kept me from believing that I had indeed made some mistake, and that crude oil had in some manner found access to my apparatus.

After many fruitless experiments I learned a fact which should have been obvious to me from the first, but which, in the surprise due to the unlooked-for result obtained, had quite passed out of my mind. In my first test, the vessel which I used had contained but a little oil (about one-fourth of the volume of the vessel only), and in all of the other experiments I had filled the vessel three-fourths full or more, in the effort to obtain as much of a yield as possible.

I repeated the first experiment, using the vessel but one-fourth full, and heating to about the same temperature, and for the same time as I had done in the other experiments. The result was once more the greenish liquid so familiar to any one who has lived in the oil fields, and its fractionation again gave 15 per cent. of gasoline, 30 per cent. of burning oil, etc.

Apparently some remarkable change must come about in the hydrocarbon molecules, when a hydrocarbon body is heated in a still approximately only one-fourth full of oil, that does not occur when the same hydrocarbon is heated under similar conditions, except that a greater proportion of the volume of the still or retort is filled with oil. With grave doubts and fears, I placed in my retort some kerosene. If this water-white material, after treatment, should come out green in color by reflected light, and red by transmitted light, then indeed I would be convinced that I was dealing with a true transformation into crude oil. The experiment ended, I poured out from the vessel a liquid which resembled Pennsylvania crude oil so perfectly that when I placed a bottle of the new product by the side of a bottle of the real crude, it was hardly possible to say which was which, by appearance alone. I next melted some paraffin and placed it in the vessel, and after heating under the prescribed conditions, I poured out a thin fluid, suggesting crude oil in every way, and which on distillation gave somewhat over 15 per cent. of a water-white gasoline, free from "cracked" odor, and other distillates in about the same relationship as in ordinary crude oil.

One after another I tried putting all natural hydrocarbons available to me through this process. Vaseline, rod wax, gas oil, fuel oil, and B. S.—all these went into my treating vessel, one after the other. They all yielded materials similar in appearance, odor, and composition. From any of these materials I obtained a synthetic crude oil containing about 15

per cent. of gasoline, and other distillates in about the same order as are found in typical crude oils.

After many experiments had shown me the exact conditions of temperature, pressure, and filling volume of my treating vessel which were necessary to success, I fondly imagined that my troubles were over. I did not for a moment think that human nature would involve greater difficulties than had even the control of natural conditions. Full of enthusiasm, I described the results of my experiments to an oil man, without of course describing the exact process, on which I had not yet applied for patent. He listened to me carefully and kindly, but his look of utter unbelief and incredulity was a trifle galling, to one whose life work had been devoted to scientific investigations. Had I been a promoter, selling stock in a gold mine located at Hackensack, or in a diamond mine on the outskirts of Brooklyn, I could hardly have met with less encouragement, or more entire disbelief.

To-day, when processes for increasing the yield of gasoline are being worked on by many investigators and when such lines of work are being encouraged and lavishly supported by a number of oil companies, and are being paid for in many cases with sums far greater than any probable returns, it may be hard for you to realize that only five years ago the shortest cut to suspicion and doubt, from your friends in the oil business, was even to suggest gently in ordinary conversation that perhaps by some method it might be possible to increase materially the ordinary yield of high-grade gasoline from crude oil. I tried it a few times, picking out the most kindly and genial of my friends in the oil-refining line. They would look, first pityingly and then suspiciously, and would say: "But after you have gotten out the gasoline that is present in crude oil, how do you think that you are going to get any more? Don't you understand that when you have gotten it *all* out why you have gotten it all? What is left then is kerosene, gas oil, or what-not. But it is not gasoline."

Only once did I venture mildly to protest. I suggested that possibly, since hydrocarbons were all compounds of hydrogen and carbon, it might be possible to rearrange the atoms in the molecule so as to obtain more gasoline. This view met with some recognition, and I was told that what I was talking about was called "cracking" and that it was thoroughly understood by oil men, and that, furthermore, "there was nothing in it" as far as making anything salable as gasoline, as the product would invariably be of bad color, and of an extremely offensive odor.

Slowly I came to realize that the oil industry was not yet ready for any new views as wholly different from the preconceived ideas as these experiments made necessary. So I would go back, for comfort, to the water-white gasoline of 70° Bé. which I had made from paraffin and from kerosene, from gas oil and from B. S., from fuel oil and from rod wax, and patiently wait for the day when my friends in the oil business would realize that

there were a few insignificant things about oil which they did not yet know. For their attitude I could hardly blame them, after all, when I remembered my own doubt when I had seen the results of my first experiments. They had not seen them, and therefore if they doubted, I could at least understand their position, and I am hardly prepared to say that I should have been less doubting than they were, had the positions been reversed.

This paper makes public for the first time the results of my experiments, and in presenting it I wish to express my indebtedness to John T. Milliken, of St. Louis, Mo., President of the Milliken Refining Co. He was the first oil man whom I met who was willing to believe that research could really add materially to the oil man's knowledge. He has generously supported the experiments which I am now reporting, and has supplied the financial help which alone has made this paper possible to-day.

It has long been known that under the influence of high temperatures hydrocarbon bodies could be thermolized, or "cracked," and that by this method low-boiling bodies could be produced from hydrocarbons of higher gravity. Indeed, the commercial use of cracking distillation in petroleum refining goes back more than half a century, the first observation of such cracking distillation being said to have been made accidentally, in a refinery at Newark, N. J., during the winter of 1861-2.

For 30 years after the discovery of methods suitable to cracking distillation, this method was in common use in many of the principal oil refineries of the world. It was found that by running the stills at a high temperature very considerably increased yields of kerosene could be obtained, and the method was found to be a profitable one from the start, particularly since in the early days of the oil-refining industry, burning oil or kerosene was the principal product sought for. The process was also investigated by many able men, and the conditions under which long paraffin chains became broken into two or more shorter chains, under the influence of high temperatures, were very carefully worked out, particularly as concerns commercial refining operations.

Accordingly it was only natural that, with the enormous increase in demand for gasoline during the past 10 or 15 years, many investigators should attempt by similar cracking methods to obtain increased yields of low-boiling products. When the vapor of kerosene or any heavier oil is passed through a red-hot tube, for example, thermolization takes place, with the production of considerable amounts of low-boiling products vaporizing within the ordinary boiling-point range of common gasoline. In this and in many other similar ways attempts have been made, both on a laboratory scale and in large-size commercial installations, to prepare products capable of replacing gasoline. Of the dozens, or even hundreds, of such efforts, few have had even the slightest promise of success, due to the fact that the low-boiling hydrocarbons produced in the manner described are off-color, and possess an odor so pronounced and disagree-

able as to greatly limit, if not wholly prevent, their sale. So acute has the demand for gasoline been at times in the past 10 years that it is not impossible that even the color and odor might have been overlooked if the process had given the large yields that were originally hoped for; but in this respect also the ordinary cracking methods have met with difficulties, and in general all these processes produce considerable amounts of tar and coke, that materially cut down their efficiency.

When the limitations of simple cracking of hydrocarbon oils at ordinary pressures were first understood, efforts were made to bring about destructive distillation under increased pressure. Results showing great improvement over those obtained by the simple cracking methods were given by these processes, which seem to have been first made use of by J. Young, and later developed by Dewar and Redwood, and others. Quite recently, improved processes of cracking distillation under increased pressures have been used commercially by Burton, and are said to have been so developed as to yield products readily salable as substitutes for gasoline.

Efforts have not been wanting to improve the color and odor of the light cracked distillates produced by ordinary cracking distillation. Treatment with sulphuric acid and alkali, in the manner commonly used in the refining of kerosene, has the effect of improving both color and odor to a remarkable extent, and by the use of sufficient acid colorless products without bad odor can be obtained, but only by the use of such large amounts of acid as to make the process commercially prohibitive, unless gasoline is selling at quite a high figure. By cracking under increased pressure the amount of acid required for this purification is very greatly reduced, and it is probably due to this fact that the motor gasoline now being so extensively developed by Burton owes its greatest commercial possibilities.

It will thus be seen that I cannot claim to be in any way a pioneer in the production of lighter hydrocarbons from materials of heavier gravity. Hydrocarbons have been cracked and broken up into lighter hydrocarbons of lower boiling point, both experimentally and commercially, for a period of over 50 years, and such cracking experiments have been conducted both at normal pressures and under increased pressures.

Other investigators have also placed hydrocarbon oils within closed vessels and have heated these oils under such conditions to elevated temperatures. In such work Engler, in particular, has made notable contributions to our knowledge of the behavior of hydrocarbons under high temperature and pressure. In these experiments it has been noted that the hydrocarbons have been broken down to lighter hydrocarbons, and that in this way low-boiling oils could be made from hydrocarbons of high boiling point. Apparently, however, the remarkable influence which is played by the ratio of the liquid contents of the vessel to the

total volume of the vessel, has been either wholly overlooked or at least not properly appreciated. It has been wholly through the investigation of the effects of the ratio of the volume of oil to the total volume of the vessel that I have developed the process which I am here describing, and which has given the remarkable and unexpected results already mentioned. I believe it is only when these suitable volume relationships are observed that we can get these results within a range of temperature and pressure adapted to commercial development.

Very careful studies made in my laboratory have now proved that when a hydrocarbon body such as gas oil, for example, is heated in a vessel which is filled to more than one-tenth of its volume with such oil, but such filling is less than one-half of the total volume of such vessel, and if then the vessel is so heated that a pressure of say 800 lb. per square inch exists within the vessel, a very remarkable and fundamental change occurs in the hydrocarbon filling such vessel. It is as though the carbon and hydrogen atoms were free to rearrange themselves, and that such rearrangement goes on until a more or less definite mixture of hydrocarbons remains in the vessel. When the vessel is less than one-tenth filled with oil considerable "cracking" seems to take place and the product is quite inferior. When the vessel is much more than one-half filled with oil the reaction seems to fail almost wholly, the amount of light products produced being very small. But when the conditions within the vessel, as to amount of filling, and temperature applied, are as indicated above, the carbon and hydrogen atoms of the hydrocarbon seem to rearrange themselves to form crude oil and natural gas.

In this rearrangement, not only are low-boiling compounds produced from those of higher boiling point, but even the reverse action takes place. In several tests I have obtained from petroleum products of medium boiling point synthetic crude oils which contained high-boiling ends, whose boiling point was considerably higher than that of any of the constituents present in the original oil used. Apparently the entire process depends upon certain equilibrium reactions, in which constituents of different boiling point tend to be present in a certain very definite ratio, provided the space relationship within the treating vessel is of the proper order. Solid paraffin of course contains no constituents that are liquid or gaseous at ordinary temperatures, but upon treatment by this process even this solid paraffin is resolved into synthetic crude oil and natural gas, and the percentage of products of each definite boiling point appear to be in a definite condition of equilibrium. If instead of starting with paraffin we go to the other extreme, and start with kerosene which is entirely free from heavy ends, we will obtain a synthetic crude oil which is much lighter in gravity than that produced from paraffin, but which nevertheless contains high-boiling constituents whose boiling point exceeds by many degrees the boiling point of the heaviest product present

in the untreated kerosene. Thus it will be seen that while this process is primarily one in which heavy hydrocarbons give crude oils containing light distillates (this being the main trend of the reaction), yet the process is so essentially one dependent upon equilibrium that if high-boiling constituents are absent, or present in very small amount, the equilibrium will not be satisfied until additional amounts of these high-boiling constituents have been produced as the result of the reaction which is going on.

A residual pressure, after cooling, always exists, due to the natural gas formed in the process, and the amount of this natural gas, like the amount of gasoline in the synthetic crude oil, seems to be very constant no matter what hydrocarbon is taken. It is of course evident to the chemist that natural gas and gasoline contain a greater percentage of hydrogen than do heavier oils, and it is very interesting to note that when the charge which is placed within my treating vessel contains a hydrocarbon deficient in hydrogen, the formation of saturated gasoline goes on just the same, and the synthetic crude oil produced carries a "mud" consisting of the carbon which in the rearrangement has failed to find hydrogen. The gasoline produced from materials even highly deficient in hydrogen is quite normal in color, and does not appear to be in any way like the "cracked" products which are produced by the thermolysis of oil vapors, etc.

The following results of runs made by this process, in one case starting with solid paraffin wax, and in the other case with Oklahoma gas oil, will clearly illustrate all the technical features of the method:

Test 1.—Material used, solid white paraffin. Melting point, approximately 120° F. (50° C.). Specific gravity, 0.925 (21.5 Bé.); 300 cc. taken. Capacity of treating vessel used, 1,100 cc. Heated until pressure of 800 lb. was indicated, then cooled. Pressure of residual natural gas, 130 lb. Product after treatment, a heavy liquid, resembling "Franklin Heavy" Pennsylvania crude oil. Color, dark green by reflected light, deep red-brown by transmitted light. Volume of synthetic crude oil obtained as result of run, 305 cc. (5 cc. increase in volume over the amount of liquid paraffin started with). Specific gravity of this synthetic crude oil, 0.770 (51.8 Bé.). Gasoline yield, on distilling this synthetic crude oil to 150° C., 48 cc. Gasoline in synthetic crude oil, 16 per cent. Specific gravity of this gasoline, 0.70 (70 Bé.). Color, water-white.

Test 2.—Material used, Oklahoma gas oil. Specific gravity, 0.850 (34.5 Bé.); 300 cc. taken. Capacity of treating vessel used, 1,100 cc. Heated until pressure of 800 lb. was indicated, then cooled. Pressure of residual natural gas, 120 lb. Product after treatment, a liquid resembling Pennsylvania mixed pipe-line crude. Color, dark green by reflected light, deep red-brown by transmitted light. Volume of synthetic crude oil obtained as result of run, 288 cc. Specific gravity of this synthetic

crude oil, 0.831 (38.5 Bé.). Gasoline yield on distilling this synthetic crude oil to 150° C., 40.8 cc. Specific gravity of this gasoline, 0.705 (68.5 Bé.). Gasoline in the synthetic crude oil, 13.6 per cent. Color, water-white.

It is of course evident that if putting any hydrocarbon through the process described makes it into a crude oil, it ought to be possible to take any hydrocarbon and first convert it into crude oil by the process described, then remove the gasoline, for example, or any other constituent, from this crude oil by distillation, and then to subject the residue to a repetition of the process. I have done this many times, and have converted paraffin and other petroleum products almost wholly into gasoline and natural gas. I have obtained from paraffin about 70 per cent. of water-white gasoline, the remaining 30 per cent. representing the natural gas formed by the repeated action of the process, and some free carbon. From fuel oil, gas oil, vaseline, and similar materials, I have obtained from 50 to 70 per cent. of water-white gasoline, and samples of this gasoline, even after standing for a year or two, do not discolor, nor acquire an offensive or "cracked" odor. I wish to particularly note that this gasoline has not been treated in any way, and has never come in contact with either acid, alkali, fuller's earth, bone black, or other related materials. In brief, the process which I have described produced, from practically any hydrocarbon, a material which resembles natural crude oil, and which gives a gasoline which appears equal in quality and appearance to gasoline produced from natural crude. Both the crude oil produced by my process and the gasoline produced from its distillation possess an odor which is somewhat different from the odor of natural crude oil and ordinary gasoline. This odor, while peculiar and distinctive, is not in the slightest like the odor of "cracked" products, and it is in fact a slightly milder and sweeter odor than that of ordinary oil products. Upon mixing my synthetic crude oil, or the gasoline produced from it, with certain muds and clays, it seems to be altered, and the odor changes and becomes much more like that due to ordinary crude oil. Personally I am of the belief that crude oil in nature has in some cases been produced by some process related to that which I have here described, the effect of the high temperature which I use for a short time having in earth history been produced by very much lower temperatures acting through geological ages. I believe the condition which in my retort is represented by about three-fourths open space, in nature has had its equivalent in the open space in the sand or other porous rock which has been the repository of the oil, and I believe that natural gas, which is so commonly associated with petroleum deposits, has had a related origin in nature to that which it has in the process worked out in my laboratory experiments.

The study of the genesis of petroleum is so involved that I do not wish these suggestions to be taken in any way as other than ideas which have

forced themselves on my mind after noting the very considerable similarity in appearance and constitution which exists in most of the petroleum of the world (except where a porous cover or other well-recognized conditions have allowed the more volatile materials to vaporize, or well-known oxidation or other phenomenon to take place), and it seems more than likely to me that any process which in the laboratory will produce materials of such similar appearance and composition from raw products of the most diverse nature, must surely have some connection with the conditions which in geological time have similarly produced, from starting-out products of many different kinds, a material possessing such well-marked and easily recognized characteristics as petroleum.

One very interesting development in connection with this work has been the effect of small amounts of certain catalytic materials in facilitating the transformation into synthetic crude oil. The addition, to the oil to be treated, of even a very minute amount of colloidal graphite reduces materially the temperature and pressure at which the process is operative. In one set of experiments, in which a given treating vessel gave synthetic crude oil at an average treating pressure of 850 lb., it was found that the addition of a small amount of finely divided graphite would lower the necessary treating pressure to 750 lb., or even to 700 lb., on somewhat longer treatment. This seems to offer confirmation of the theory which I have advanced, that the entire process is dependent on certain reversible reactions which under the described conditions reach an equilibrium when sufficient time is given. The action of finely divided catalytic materials in increasing the speed of reactions is well known, and in these experiments their function seems largely to be the increasing of the rapidity of the equilibrium reactions, so that the reaction goes on more nearly to completion in the very brief time of the test. In my experiments our general procedure has been to heat the treating vessel until the desired pressure is indicated, when the heating is at once stopped, and the treating vessel cooled and emptied. We have found that when, instead of raising the pressure to the desired treating maximum, and instantly cooling the vessel, we raise to a somewhat lower temperature, and maintain this temperature for 5 or 10 min., we get practically an equivalent result. Where a catalyst is used, as described, it is possible to use a much lower pressure and still obtain a normal synthetic crude oil.

While the process has been described in this paper as an intermittent operation, the underlying principles being much more easily understood in this form, it is of course evident that the process is capable of continuous operation, and in many of my tests I have made use of apparatus adapted to such continuous transformation of heavy hydrocarbons into "synthetic" crude oil. In these experiments the heavy oil taken is forced continuously into the treating chamber, which is maintained at a suitable working temperature, and the synthetic crude oil and natural gas

are continuously removed, under such conditions as maintain within the treating tube the proper relation in regard to charging volume. The synthetic crude oil produced in this way may be distilled in the usual manner, and its gasoline content removed and the residue re-treated, or by comparatively slight modifications of the process, the synthetic crude oil may be fractionated within the treating vessel, so that light hydrocarbons alone reach the discharge tube of the apparatus.

These experiments which I have described have been wholly of a laboratory nature, and much work remains to be done in the application of the principles which have been discovered to commercial work on a large scale. While it may seem to many that the pressures and temperatures employed are so high as to preclude the possibilities of commercial work, yet I do not think this is the case. Processes have been developed abroad, during the past few years, in which ammonia is made synthetically by reactions requiring both higher pressures and higher temperatures than those which are made use of in my present work. As these ammonia researches have gone on from their laboratory inception to their commercial development upon a very extensive and successful scale, I believe the present process will find similar development comparatively easy. The conditions necessary for successful commercial work are already well known, and involve no engineering features which American ingenuity cannot easily provide, and it is my hope that this process will be soon developed to the point where it will fulfill commercially the remarkable promise that it now seems to offer.

Biographical Notice of Samuel Benedict Christy

BY R. W. RAYMOND, NEW YORK, N. Y.

(Reprinted with some additions and changes from the *Engineering and Mining Journal*)

THE death of Prof. Samuel Benedict Christy on the 30th of November, at the age of 61 years, cuts short a brilliant and influential professional career. He was born in San Francisco, Aug. 8, 1853, and was graduated as Bachelor of Philosophy at the University of California in 1874. For the five years that followed, he studied mining and metallurgy as a post-graduate in the same institution, serving also as instructor in analytical chemistry, and becoming in 1879 instructor in mining and metallurgy—a position which he held until 1885. During this period he became known also by his contributions to technical literature, the earliest of which, perhaps, was his report on the Monte Diablo coals (in 1875); but the first which drew my attention to him was an exceedingly able discussion (in 1879) of the genesis of the quicksilver deposits of California, in which he followed with much acuteness and originality the lead of Professor Becker, who had been teaching at the University of California. The notion of the thermo-aqueous formation of such deposits was then new; and the laborers in a field now familiar to every student deserve the credit due to pioneers.

For some years the subject of quicksilver seems to have occupied much of Christy's attention. In 1877, he described the mines and works at Almaden, Spain; in 1884, the Imperial quicksilver works at Idria, Austria; in 1889, the New Almaden mines of California.

But his great work was the creation of the school of mining and metallurgy at the University of California, which may be said to have begun with his occupancy of a full professorship of these branches. It is not necessary here to recount the early difficulties and perils of the University, and its long struggle for academic independence of political control or interference. The battle is over, and the institution is now upheld by the intelligent loyalty of the citizens of California, without regard to party. But while the issue was uncertain, the development of special schools of applied science was scarcely practicable. Doctor Becker and William Ashburner had given up the school of mines as a hopeless task when Christy, younger and less experienced, took it up. As he wrote me in February, 1901—

"I took an empty building without equipment, and with half a dozen

students. Now I have a fully equipped department, adapted to our local needs better than any I have seen in the world. I have now nearly 250 students; and I am willing to compare the record of our graduates, now scattered all over the world, with that of any other institution of equal age."

The secret of this success lay in the enthusiasm, pertinacity, industry and personal charm of Christy himself. He was able not only to inspire his students, but to impress upon their parents and the public at large the value of the thorough training given them under his direction. His own example was contagious. Besides the practical labors of his office, he prosecuted laboratory researches of the most delicate and complicated character, in which it was a privilege for any student to be his assistant.

Before the famous Hearst building was erected and equipped for the School of Mines, Professor Christy traveled through Europe, inspecting the great laboratories and plants of technical instruction. By that time, his work had made him widely known; and I remember still with what simple-hearted surprise he expressed his gratitude for the reception he had encountered abroad.

I would quote in this connection, the peroration of his address at the dedication of that building, Aug. 23, 1907—a day which marked the summit of his life, and an address which reveals the heart of his high ambition:

"Deep-rooted in the eternal hills, this memorial to Senator Hearst, in simple dignity, beauty, and strength, lifts its noble head into the luminous air. Hewn from the solid granite of our own Sierra Nevada; molded from the plastic clay of our own valleys, and bound together with bonds of steel; designed with consummate skill, and executed by hands that loved their work; it went through the great earthquake absolutely unscathed, as if Nature herself had marked it with her approval.

"And now, may I not endeavor to express the profound sense of gratitude that fills every heart; the deep feeling of responsibility of those who accept this sacred trust; and their unfaltering determination that the men who go forth from these walls shall be worthy of their high calling and of their great opportunities!"

Yet for many years this already illustrious teacher, investigator, organizer, and leader, who was delivering annually to scores of young graduates their diplomas as mining or metallurgical or chemical engineers, had held no degree himself, except the primary baccalaureate in philosophy, acquired before he began his technical course. The reason was that in 1879, when he finished that course, the University of California was not yet granting technical degrees.

In 1902 Columbia University put an end to this anomaly by conferring upon the head of the California University School of Mines—one of the



Samuel Benedict Christy

foremost rivals of its own school—the honorary degree of Doctor of Science. This graceful recognition was creditable alike to giver and recipient, and it was a great joy to Professor Christy, after his long and arduous labors in a distant field, to be thus received into the academic brotherhood. He traveled extensively at that time in the Eastern States and the Lake Superior region, studying everywhere with keen observation the methods and appliances of mining and metallurgy. In June, 1902, just after he had received his doctor's gown, he wrote me:

"We went to-day through the subway with Mr. Parsons and examined the part where the landslide took place. They secured the mining ground in a most original way—ran in several thousand barrels of cement grouting—and are now drilling and blasting out the cemented ground. Verily, you do wonders here in New York."

Professor Christy became a member of the Institute in 1883, and was a Vice-President in 1891 and 1892. His contributions to the *Transactions* were:

Title	Year	Volume
The Miners' Fund of New Almaden.....	1884	XIII
Quicksilver-Reduction at New Almaden.....	1884	XIII
Quicksilver-Condensation at New Almaden.....	1885	XIV
The Losses in Roasting Gold-Ores and the Volatility of Gold.....	1888	XVII
The Growth of American Mining Schools and Their Relation to the Mining Industry.....	1893	XXIII
The Solution and Precipitation of the Cyanide of Gold..	1896	XXVI
The Electromotive Force of Metals in Cyanide Solutions.	1899	XXX
Biographical Notice of Joseph Le Conte.....	1901	XXXI
Present Problems in the Training of Mining Engineers.	1905	XXXVI

Of this list, the remarkable notice of Le Conte has been generally recognized as the most appreciative and satisfactory of the tributes paid to that illustrious and beloved leader and teacher.

At the San Francisco meeting of October, 1911, he read an elaborate paper, illustrated with lantern slides, on The Electro-Deposition of Gold and Silver from Cyanide Solutions, which presented in detail the results of his marvelously minute and patient researches in that field. Unfortunately, the manuscript was never furnished for publication.

But a man so intensely and continuously active in so many directions leaves "foot-prints on the sands of time" too numerous to be traced and interpreted by a hurried reporter. In addition to what has been already mentioned, I know that Professor Christy discussed in 1891 the practice of chlorination at the Alaska-Treadwell mine; that he was engaged for a series of years in the U. S. Circuit Court as an expert in the metallurgy of quicksilver and of lead; and that in 1900, as the result of patient and thorough experiment, he patented an improved process for the recovery of gold and silver from dilute cyanide solutions. But

beyond this record, which is enough of itself to secure for him a permanent place in the history of American scientific and technical progress, there remains his work in connection with the California Academy of Sciences (of which he was a life member, and for five years Corresponding Secretary), the Society for the Promotion of Engineering Education (of which he was three years Vice-President), the Mining and Metallurgical Society of America (of which he was a charter member, and Chairman of the San Francisco Section), the California Miners' Association and other bodies.

I cannot altogether forbear, though I hesitate, to speak of more sacred things—of his marriage in 1881 and the companion who is left desolate by his departure; of the anguish which they suffered in the death of a son on whom their hopes were set; of the close sympathy into which I was then drawn, by reason of my own similar bereavement, and of the unbroken loyal friendship, both before and after that event, which time did not, and death cannot, impair. I rejoice that a quick and peaceful end has crowned a fruitful life, and that the bright, strong spirit of my dear friend, "crossing the bar," went out to sea through the Golden Gate he knew and loved so well.

Some Problems in Copper Leaching

A Discussion at a Meeting of the New York Section, Jan. 6, 1915.

L. D. RICKETTS, New York, N. Y.—In recent years the metallurgical field of the copper industry has expanded greatly, the copper ores have become lean and diverse in character, and we are obliged to treat such ores on a very large scale.

On the commercial side, the operating and consulting staffs of the great mining companies have been enlarged and organized, and no great capital expenditure is decided upon until it has been investigated and recommended by a highly organized force of specialists as a committee.

I believe that in the scientific field and in the most valuable proceedings of our various societies the old idea of one-man point of view and one-man treatment of principle and detail together has been splendidly developed and should continue, but for the advancement of our art we should deliberately discuss subjects among ourselves in order to do away with the idiosyncrasies of the individual and to gain a composite view representing the best thought of many men.

In the development of new ideas leading to the introduction of new methods, the societies could enlarge their present great field of usefulness if such discussion be had, and in my opinion further great good could be done if a number of men of different qualifications and experience would join in preparing a joint book, each dealing with his particular problem, but consulting the other men. In this way individualism would be preserved and various processes would be described, and if properly reviewed the paper would be palpably capable of analysis, and the student would have at his command the associated efforts of a group of qualified engineers.

I have neither the time nor the qualifications to make an investigation and study in detail of hydro-metallurgical processes, but for some years I have foreseen the development of such processes, and I have a deep interest in the subject and have given it in some of its phases careful consideration from a long-distance viewpoint. I have assisted in directing many experiments and in reviewing the merits of many processes, and am glad to make some introductory remarks with the understanding that other gentlemen present will freely and candidly enter into the discussion and help to fill out the few bare bones that I can uncover. You will see in my remarks that I am controlled very largely by operating demands. No matter how beautiful the process, simplicity and cheap-

ness of operating costs are the main features that appeal to me, and complexity and increased costs only appeal to me if they definitely promise an increased yield that will more than offset the increased expense; for, given the grade, the maximum profit per ton is demanded.

As you all know, there has been a very large development of deposits of lean copper ores in the Southwest in recent years, and this region has rapidly developed until in the aggregate it produces more copper than any like area in the world, and it still has a large increase of output ahead of it. The vast mass of these lean deposits consists of sulphide ore, and until recently these sulphides alone have attracted serious attention. It is well known, however, that there are large tonnages of oxidized copper ore associated with the lean porphyries.

A few years ago the Calumet & Arizona Mining Co., upon the recommendation of J. C. Greenway, took an option upon a majority of the stock of the New Cornelia Copper Co., whose mining properties are at Ajo, in southwestern Arizona. Mr. Greenway directed systematic drilling upon this property, followed by the sinking of some 70 shallow shafts for test purposes and the sinking of two deeper shafts, with considerable underground development, for the purpose of testing and checking the grade of the sulphide ore. The stock was purchased.

The matrix of the ore is an eruptive granite. There is no overburden. The chalcocite has been oxidized nearly as fast as formed. The zone of transition from oxidized minerals to primary sulphides is narrow. The oxide and sulphide ore is remarkably uniform in grade. Ira B. Joralemon's paper, in the *Transactions*,¹ describes this deposit. He shows that there are about 40,000,000 tons of ore containing $1\frac{1}{2}$ per cent. copper, and the tonnage is divided into about 12,000,000 tons of $1\frac{1}{2}$ per cent. oxidized ore and 24,000,000 tons of sulphide ore, with about 4,000,000 tons of mixed oxide and sulphide between. If the grade be dropped to an average of less than $1\frac{1}{2}$ per cent., these tonnages would rapidly increase.

In parallel with this development, oxidized ore in notable tonnages have been found in the Globe mining district. This ore is more scattered, and, on the whole, is leaner than the Ajo ore, and frequently has a large capping of overburden. In the aggregate, the Inspiration Consolidated Copper Co. alone now has over 15,000,000 tons of oxidized ore running about 1.3 per cent., and a probable tonnage in addition. There are other mines in other districts containing oxidized ore, notably in Utah and Montana, and there are also oxidized ores at other points in Arizona and a vast deposit in Chile. The treatment of such tonnages is of prime importance.

The Ajo ore has a matrix of granite with a large percentage of sec-

¹ *Bulletin* No. 92, August, 1914, pp. 2011 to 2028.

ondary quartz replacing feldspar. There is little calcium present in soluble form, and the oxidized ore seems to be best adapted to a leaching process. Preliminary laboratory tests showed that the ore crushed with the production of remarkably little slime and that the copper will dissolve in dilute sulphuric acid quite freely when the crushed ore contains fragments no larger than 6-mm. cubes. Tests also showed that a little cuprite is present, which is only partly soluble in sulphuric acid. There are also other soluble salts, notably salts of iron and aluminum. It was decided to investigate a leaching process for the oxidized ore, and since the ore made but little fines and could be leached with comparatively coarse crushing, we decided that we would try to leach in tanks by percolation. The amount of silver in the ore is small. Salt is apparently expensive, and it was decided that, other things being equal, the use of chlorine compounds as solvents was not desirable. I know, of course, that chlorine compounds may have a great future in leaching, but in this particular case, and in the early development of a process, it was considered that sulphuric acid offers fewer difficulties as a solvent.

To me the hydro-metallurgical problem divides itself naturally into two nearly separate divisions—the solution of the copper, and the recovery of the copper from solution. We were obliged, however to study the problems of solution on the basis of getting the copper into solution in the form most convenient for its ultimate recovery.

At the start we employed Stuart Croasdale to undertake a study of the solution of the copper, and we also delegated to him experimentation upon the precipitation of copper by iron. Mr. Croasdale ably conducted a long series of most valuable experiments and published a paper upon this subject which you will find in the *Transactions* of the Institute.² At the same time we were fortunate enough to obtain the advice of Uteley Wedge upon the manufacture of sulphuric acid direct from calcining-furnace gases.

Mr. Croasdale demonstrated that the ore crushed to an 8-mm. cube, or smaller, would yield over 80 per cent. of its copper when treated with a dilute solution of sulphuric acid. He demonstrated that slime presented no difficulty if the ore was distributed in the tanks uniformly, and that by finer crushing 85 per cent. of the copper could be obtained in solution if treated for a sufficient length of time with an acid solution. He demonstrated not only that the ore was permeable uniformly, but that with comparatively small tanks the height was not a matter of great consequence, and he was easily able to use a column 12 ft. in height, and could probably use one much higher. Later on we found upward percolation preferable.

Mr. Wedge showed us that with care in calcining the sulphide fines

² *Bulletin* No. 92, August, 1914, pp. 1881 to 1929.

for reverberatory furnaces at the Calumet & Arizona smelter at Douglas, sulphuric acid of 55° to 60° Bé. could be manufactured with a low operating cost, and a construction cost of not to exceed \$3,000 per ton of daily capacity. The point of manufacture is about 300 miles, more or less, according to the route, from the mine. Mr. Wedge also undertook a study of sulphating and chloridizing roasts. His requirements were that the ore be crushed to pass a 1½-mm. screen. He showed that if the sulphide ore could be mined in parallel with the oxidized ore, and in proportion to their respective tonnages, there would be sufficient sulphur for his purposes and that a very high extraction could be obtained. He also showed that sufficient advantage could not be had in using salt to offset its cost and the difficulties that might arise later. Mr. Wedge also showed that a small amount of heavy sulphides from the Bisbee mines crushed and mixed with the oxide ore would give a very high percentage of extraction after the mixture had been subjected to a sulphating roast. He also introduced the idea that a preliminary water leach would give a perfectly pure sulphate solution from which the copper could be recovered electrolytically and that the acid thus regenerated would be sufficient to complete the leach.

Frederick Laist in the meantime was conducting a number of experiments governing processes in a like brilliant way, and among these processes was the development of an oxidizing roast on Anaconda tailings. Like Mr. Wedge, Mr. Laist found that by calcining sulphides or mixed sulphides and oxidized ores a high percentage of the copper could be extracted by leaching. Temperature control was vastly important, as the developments of both Messrs. Wedge and Laist showed. If too high a temperature is used a large amount of the copper becomes insoluble in the strongest acid, but in both cases temperature control was demonstrated to be simple and practicable, and under these circumstances practically no insoluble copper compounds were formed. In Mr. Laist's case, however, the use of salt is essential on account of the silver contents of the charge, but he finds that it is not necessary to use this in a chloridizing roast and that a solution of salt with sulphuric acid gives him the desired extraction on the product from the oxidizing roast.

The manufacture of sulphuric acid by the chamber process, while simple, requires a very large investment, and in this case the transportation charges would amount to more than the cost of the acid. The chamber and the contact processes, of course, are the two important manufacturing processes, but even a casual study of wet methods of copper recovery shows that there are other methods of acid manufacture involving a principle similar to the chamber method, that might be used in parallel with the recovery of the copper from solution with less investment for plant and with a saving in energy that is important.

The McKay process includes a sulphating or an oxidizing roast, as

he chooses, to get sulphide of copper into soluble form, and he adopts the same process for this purpose as followed by Messrs. Wedge and Laist.

The Slater process and the Midland process deserve study. They depend upon the use of complex series of chlorides and hypochlorous compounds that will dissolve many salts of copper that sulphuric acid will not touch. They will attack, among other things, chalcocite, and they deserve serious consideration in many important but special cases, as they avoid the necessity of a preliminary roasting operation.

I regret that I am only slightly familiar with the very important work in the development of a process by E. A. Cappelen Smith for the beneficiating of the Chuquicamata ores.

RECOVERY OF COPPER FROM SOLUTION

At the beginning of our investigation we seriously considered what I called the "brutal method" of leaching, namely, the manufacture of sulphuric acid, the solution of the copper from the ore with such acid, and the precipitation of the copper by metallic iron, with the resultant complete, or nearly complete, destruction of both acid and iron and the production of an impure cement copper that will have to go through the process of smelting and refining. The results of Mr. Croasdale's tests and Mr. Wedge's cost figures indicated that such a course was commercial, and we concluded that the $1\frac{1}{2}$ per cent. Ajo ore, with no overburden, mined by steam shovel and crushed coarse, would yield a profit on a 12c. copper market if a cheap grade of Alabama pig iron were used.

But it occurred to me, as it had occurred to many engineers, that oxide of iron could be reduced to metallic iron without fusion, and if so we might bring Bisbee sulphides to the mine, calcine them for the manufacture of sulphuric acid and then metallize the available iron in the impure calcine, already containing values requiring recovery, and use this for a precipitant. I found at the start that metallic iron was being produced by this method with an inferior fuel from high-grade iron ore by an intermittent process; that is, by first heating to the desired temperature by internal combustion of a part of the excess coal in the charge and then allowing the heated charge to stand without the admission of air, but allowing the gases formed to escape freely.

After consultation with friends, we concluded to try the experiment of a multi-hearth furnace, and at first my idea was that possibly calcining could be done on the upper hearths and the hot calcine could then pass to the lower hearths, where it could be further heated and submitted to a reducing action. We erected a small Wedge furnace, but only tried to metallize, and not to calcine in it. Some of the iron oxide was reduced to metallic iron, but on the whole our experiment was a

gas produced was sufficient to retard and reverse the action of carbon monoxide in reduction. In parallel with this, we also made experiments in revolving cylinders, aiming to reduce the iron to the metallic form with illuminating gas and with fuel oil by a continuous process. These also were failures.

Mr. Laist conducted similar experiments on a continuous process for making sponge iron at Anaconda, and ended with a like failure. He has, however, demonstrated that he can make sponge iron by an intermittent process in a revolving drum. It had become evident to me, even in my limited environment, that the simple use of iron, with the destruction of both iron and acid, to the production of an unfinished product, while apparently commercial, was in our extremely simple case preposterous, and investigation showed the splendid work done on other lines and convinced me that if considered from a commercial viewpoint better methods were bound to develop a process that would fit our requirements, and very much cheaper copper would result.

At the start we decided to investigate the electrolytic deposition of copper from solution, but a little later on we took up the subject of the direct precipitation of the copper by sulphurous acid gas. I prefer to attack the latter subject first for the purposes of this paper.

In the precipitation of copper by sulphurous acid gas, temperature and consequent pressure must be considered. If the copper is precipitated in metallic form this temperature and pressure are very high, but if a sub-chloride of copper would satisfy us, the temperature and pressure may be moderate. In both of these processes, the reactions we are familiar with in the formation of sulphurous acid by treating copper turnings with concentrated sulphuric acid are reversed by heat. For the production of metallic copper the action depends upon getting sulphurous acid dissolved in the sulphate solution in sufficient amount to precipitate the copper and then heating the mixture under proper control to about 150° C. The question of the use of an abnormally rich gas is involved. Under proper regulation pure copper is produced, which only needs melting for the market, and a large excess of sulphuric acid is formed. Mr. Van Arsdale has done valuable original work on this process, followed by the work of Weidlein and others in its commercial development. Laist has studied precipitating sub-chloride of copper. A soluble chloride is added to the sulphate solution of copper and the proper amount of sulphurous acid is added. Moderate heat under pressure— 90° C. of heat—precipitated almost all the copper as sub-chloride. The subsequent process of obtaining the copper without volatilization and the recovery of the chlorine offered difficulties, Messrs. Van Arsdale and Bacon have valuable experience on this subject.

These processes need further development and are most interesting

It is with no disparagement of the process that we have abandoned it in our case. I simply feel that the method would be a complicated and an expensive one, and I fear working with heat and pressure where fuel is dear and a corrosive solution is used. To me and my friends the electrolytic method seemed best adapted to our condition and the simplest and more certain of commercial application.

Of course, the first difficulty was the solution of other oxides than copper and the danger of some loss of acid thereby, and, what was more important, the susceptibility of some of these dissolved neutral salts to oxidation by nascent oxygen to the production of acid salts, which in turn will dissolve copper from the cathode. It seems that such salts are most valuable as depolarizers, but must be held under control.

After investigating the subject I advised Mr. Greenway on behalf of his company, and we employed F. L. Antisell to make and direct a series of tests and experiments covering the precipitation of copper in the presence of iron and aluminum sulphates. For some time previous I had been familiar with experiments tried by Messrs. Pope and Hahn, and these gentlemen were employed to investigate in parallel the practicability of removing enough of the iron and alumina from the solutions as they foul to keep them below the danger point in electrolytic work. Both Mr. Antisell and Messrs. Pope and Hahn began their experiments upon a laboratory scale at the Raritan Copper Works. Messrs. Pope and Hahn found the leaching could be so conducted that the rate of increase of iron and alumina was slow. They therefore decided to leach in a certain way and to treat a certain high iron and aluminum neutral solution with copper oxides by agitating, heating, and injecting air. This is a rejuvenation of the old Hoffman process applied to a new purpose. The iron and aluminum sulphates in such foul solution, together with arsenic and other impurities, are precipitated as insoluble oxides and the acid previously combined with these salts is taken over by the copper oxide and forms sulphate of copper. The clarified solution, with these impurities removed, is then mixed with the balance of the solution, thus keeping an iron content of, say, less than $\frac{3}{4}$ per cent.

Mr. Antisell began his experiments by using a patent anode. This anode was a narrow, rectangular wooden or lead frame with thin wooden sheets as diaphragms, forming a long and deep but very narrow box. It was packed with a clean coke in lumps crushed to pass a 1-in. ring, and contact was given by carbon rods imbedded in the coke. A solution of sulphate of iron and aluminum in great excess was made up and sulphuric acid added. This was used as a solvent for the copper. After the solution came from the ore it was at first passed to an absorption tower, where sulphurous acid gas was absorbed, and then it passed to the electrolytic tank. I go into no detail in describing Mr. Antisell's flow sheet and his process, as he himself is present. After promising results had

been had in the laboratory tests at Raritan showing the possibility of high extraction and low power costs, Mr. Antisell sent Mr. Jamieson to Douglas, where tests were made with full-sized anodes. Excellent results were obtained here, but difficulty was had with the complete absorption of sulphurous acid. Also, the anode was cumbersome and not adapted to operating conditions, but, notwithstanding, excellent results were obtained experimentally and he ran for a long time with a recovery of nearly 2 lb. of copper per kilowatt-hour, an extraction of 80 per cent. of the copper contents of the ore, and the production of nearly the theoretical amount of sulphuric acid from the SO_2 gas.

Messrs. Pope and Hahn later went to Douglas and began experiments, and also later conducted a most valuable set of experiments on the manufacture of acid-proof coatings for concrete that will neither crack nor soften under the high temperatures of the Arizona desert, and which will adhere closely to the sides of the tank. This work is very important and needs mention.

Mr. Greenway discovered that if a certain copper solution containing salts of triad elements is treated in a simple way, the iron or aluminum triad salts may be removed. Dyad salts are not affected. He was led to the belief that such oxides thus precipitated were in large part insoluble in dilute acid and that therefore he had a very simple method of removing iron and aluminum from solution. Against my prediction, laboratory tests seemed to show his contention was correct. Having used up about 300 or 400 tons of Ajo ore at Douglas for the Croasdale, Antisell, and Pope and Hahn experiments, and having promising results, the engineers moved to the mine at Ajo. Mr. Greenway erected a small plant of his own, and a Pope and Hahn plant was installed and the use of sulphurous acid was temporarily discontinued. As for the Greenway process, I am not yet at liberty to discuss it further. But I may say over 100 charges were run with this process. The iron content in the solution was kept down to less than 1 per cent., and we obtained an average of slightly over 1 lb. of copper per kilowatt-hour, but the extraction was low. We then extended the time of the leach and obtained a higher extraction, and finally we began to crush the ore, which at that time had been crushed only to $\frac{1}{2}$ -in. ring, finer, and found that if crushed to pass $\frac{1}{4}$ -in. mesh percolation was not retarded and an extraction of over 80 per cent. was practicable. The iron has reached as high as 1.2 per cent., but we are still getting a pound of copper per kilowatt-hour from the electrolytic tanks.

These experiments have been so successful that we are proceeding with the erection of a 40-ton plant, where tanks will be used that will hold 60 tons, if necessary, and will be abundantly large to permit of a study of the comparatively even distribution of crushed ore, and we shall proceed with these most interesting experiments, using first, as we have

in the smaller tanks, the Greenway process. In the meantime, the 1-ton plant will be liberated and we will immediately proceed with the continuance of a modified form of the Antisell system of leaching. He finds that with the construction of better absorption towers we can absorb the necessary amount of SO_2 gas, and he has demonstrated that with the use of SO_2 gas, using dyad iron as a depolarizer and triad iron as an oxidizer, he can manufacture sulphuric acid. We can, if successful, obtain $1\frac{1}{2}$ lb., and possibly more, of copper per kilowatt-hour, and at the same time manufacture sufficient acid to replace wastage. The electrolytic copper in all cases has been fully up to the trade standards.

CONCLUSIONS

My deductions from a study of the work of the able engineers engaged in these extensive operations are as follows:

1. Dilute sulphuric acid will give a very high extraction on oxidized ores of the Ajo type. Time is a more important element in extraction than strength of acid. A maximum size of 6-mm. cube is required. Such ore crushed to the maximum size permits percolation and the production of a clear solution because the slime content in the ore so crushed is very small.

2. Where such ore is mined at a cost of between 25c. and 50c. a ton, it is commercial on a 12c. market, even if leached with sulphuric acid and the copper precipitated with pig iron to the waste of the acid and iron and the production of an impure cement copper that requires further treatment. Metallized iron from calcines can be made and will prove a little cheaper than pig iron if it is marginal.

3. An electrolytic method for the recovery of pure copper from sulphate solutions is far preferable and will produce cheap copper. In all cases a low current density is required. Hard-lead anodes or composite anodes of hard lead and coke of uniform thickness will be used.

4. Such electrolytic method, with control of the amount of iron and aluminum in the electrolyte, and without the use of sulphurous acid gas, is practicable. An acid loss of about 1.5 lb. per pound of copper will be sustained. Chamber acid will be required.

5. I believe there is a decided probability that the cheapest and best method will be the electrolytic deposition from sulphate solutions without iron control, and the use of the requisite quantity of sulphurous acid. The sulphurous acid will be absorbed without the tank house and in quantity that will not cause annoyance. Iron and aluminum will act both as depolarizers and catalytes. This process promises a high yield of cathode copper per unit of power and the manufacture of sufficient sulphuric acid to replace the waste.

6. While these remarks apply to the Ajo oxidized ore and ores physic-

ally and chemically similar, they need not apply to ores of a different composition. Mixed oxide and sulphide ores, be the sulphide chalcocite or chalcopyrite and bornite; ores containing great quantities of clay, or presenting other difficulties, have to be considered separately, and there is beyond question a great field for other processes and for other methods, both for obtaining the copper in solution and obtaining the copper from the solution.

And now, gentlemen, while thanking you for your attention, I end my theme as I began. Notwithstanding the importance of this subject, it covers but a corner in the metallurgy of one metal. There are many other subjects which if discussed in committee and with an open mind will lead to the advance of ourselves and of our institutions.

UTLEY WEDGE, Ardmore, Pa.—It has certainly been very gratifying to hear such a comprehensive discussion of the development of the technique of leaching in connection with the property of the New Cornelia Copper Co. The presentation of this matter in the paper as read I believe to be accurate, but one aspect of the problem has not been presented. The paper relates almost exclusively to the treatment of carbonate ore. The deposit at Ajo, where this method is to be applied, comprises about twice as much sulphide ore as carbonate ore. It also comprises 1,500,000 tons of ore which is a mixture of sulphide and carbonate ore. It would be interesting to have accurate data as to the yield or percentage recovery of copper that can be secured from the sulphide ore by the various methods of concentration. The character of the ore leads me to believe that the percentage of recovery of values from the sulphide ore by any of the wet concentration or flotation processes in use at the present time, will be materially less than the percentages which this paper has shown us it is possible to recover by a leaching process. In considering the ultimate beneficiation of this property, I would suggest that the treatment of the carbonate of copper ore should always be considered in connection with the treatment of the balance of the deposit.

Is it not true that the sulphuric acid leaching plant for carbonate ore as described, if constructed on a large scale, will have largely outlived its usefulness when one-third of the deposit of ore has been worked out? If the same facilities that will handle the carbonate ore would also handle the sulphide ore, so that the same plant and the same processes could be utilized throughout the operation, the eventual total cost of construction might be materially reduced. The leaching of carbonate ore of this character with sulphuric acid is clearly demonstrated to be a success. It is clearly demonstrated that the carbonate ore which is to be leached with sulphuric acid does not need to be finely ground to leach completely and economically. It is apparently demonstrated that with processes recently perfected SO_2 can be introduced into an impure

electrolyte with advantages in economy as regards consumption of power in electrolysis and quality of product, but these advantages can largely be retained by the following operation:

Let the sulphide ore and the ore which is found between the mass of carbonate and the mass of sulphide ore, which partakes of the character of both, be ground to 20 or 30 mesh and furnaced at a temperature not much above a red heat, and the material can at once be leached with as good percentages of recovery as have been secured in the case of the carbonate ore. The copper solutions secured can be electrolyzed, and sufficient sulphuric acid can be produced from the electrolysis for use in leaching all or a portion of the carbonate ore, which methods can be identical with the methods described in the paper.

The leaching of the finely ground sulphide ore will call for somewhat modified appliances on account of the fineness of the material, but it has been abundantly demonstrated in other plants and in other camps that finely ground material can be economically leached. There are methods of handling finely ground material in leaching which even seem to have advantages in operation as compared with the methods described for leaching material in lumps.

In case the leaching plant projected, as indicated by the paper read, is built no larger in capacity than would in any event be necessary in connection with the treatment of sulphide ore and carbonate ore by the methods indicated, then and to that extent the construction for such acid leaching of the lump material would be serviceable for the life of the property, if the construction is of such character as to last that long, and if the mining and beneficiation of the carbonate ore can be so arranged that it will extend over a like period.

I am of the opinion that sulphatizing roasting of the sulphide ore in the orebody of the New Cornelia Copper Co., coupled with the acid leaching of all or a portion of the carbonate orebody with sulphuric acid by the methods indicated in the paper, will give the lowest possible cost of operation and will give the highest possible recovery of values of any method known to the metallurgical art. This combination of methods would obviate the necessity of construction of a sulphuric acid plant; in the place of this plant there would be constructed a furnace plant for giving the required roast to the sulphide ore. At the start, with only carbonate ore accessible, enough of the carbonate ore could be roasted mixed with small percentages of iron pyrite from neighboring camps to sulphatize the carbonate of copper.

I believe the cost of a roasting plant for conducting the process in this way is amply justified by the large recovery of values which can be made, and I believe that the advantages of this combination of methods are so great that the beneficiation of this property by such methods would yield several million dollars in excess of what would be secured if the sul-

phide ore were treated by any of the known concentrating processes followed by the smelting of the concentrate.

As a general statement, the class of copper ores known as oxidized ores and carbonate ores, to the extent that they are free from alkaline earths or other gangue materials that are readily attacked by sulphuric acid, will, in general, lend themselves to the treatment that Dr. Ricketts has described: namely, the treatment with sulphuric acid. But the past history of the leaching method as applied to the metallurgy of copper is full of instances where efforts have been made to accomplish results by leaching processes not preceded by the study which was given to this matter in connection with the New Cornelia operation. Such attempts in the past have justified the criticism of leaching processes which was once made in a general way by the father of a gentleman whom I see present, who said that the chief thing he had against leaching processes was that they were so *Eine Schweinerei*. If I am asked to give the English equivalent of that, I would say that it is apparent that the installation of leaching processes which the German gentleman had seen did not appeal to his esthetic sense.

But another day is very rapidly approaching, and the advance that has been made in the past few years in this line of progress seems to me to be revolutionary. There is so much being done by so many different companies that the efforts of any individual in connection with such practice must seem small.

The connection that I have had with this matter had led me to emphasize especially the effort to secure pure solutions which would yield to electrolysis without difficulty; and the development which has been brought about by two gentlemen who will be heard from this evening makes it seem probable that solutions quite impure in character will hereafter be economically electrolyzed. I regard the work of these gentlemen as of the utmost importance. The application of leaching processes to copper ores and copper calcines is practiced on a larger scale than is generally known. Over 1,000,000 tons yearly are treated for the extraction of copper by leaching. The largest leaching operation in the world is one operation which treats about 750,000 tons per annum. I refer to the Rio Tinto Co. The process used by the Rio Tinto Co. is in its essential character a sulphatizing roasting process. The ore is piled up in heaps 30 ft. deep, which are called *terreras*, and the ore is allowed to heat by chemical action until a sulphatizing roasting condition is produced and sulphates of copper are formed, which are leached out. The balance of the leaching of copper ores, done on a large scale throughout the world, is mostly chloridizing leaching, where salt is used. The leaching process with salt is very extensively carried out in Germany and England. There are several hundred thousand tons, I think, chloridized every year in Germany and England. I know individually

of quite a number of very large operations there with which I have come in contact in connection with the installation of furnaces for chloridizing roasting. There is a very considerable tonnage chloridized in this country. The chloridizing process is familiar to a great many here. For the benefit of the others, I will say that in general, ore, which has previously been roasted to deprive it of most of its sulphur, down to a point where the sulphur is from one and one-quarter to one and one-half times the copper content, is mixed with a percentage of salt, usually in good practice in the neighborhood of 10 per cent., and this mixture, which is reduced to a fineness of 3 mm. by grinding, is then heated in chloridizing furnaces to a dull red temperature, or even slightly below, and the copper is chloridized and subsequently leached out with water, and with such hydrochloric acid as is caught in the scrubbing towers in connection with the furnaces.

The use of sulphur dioxide in connection with the precipitation of copper is a matter which personally I have been watching with very great interest. The quantity of sulphur gas which is escaping from the different smelter plants in this country is so prodigious that it seems like a waste; it seems as if this material will eventually be utilized and turned to some account. It is very possible that to some extent this will be brought about in connection with either the electrolytic or other precipitation of copper, where sulphur dioxide is evidently going to be very much in demand. There has been a possibility, if not a probability, that liquid SO_2 might be required in this connection. It is interesting to note that at the present time, as far as I know, there is no liquid SO_2 plant in the United States. There are several in Germany, and one or two in England, and the stoppage of the importation of liquid SO_2 from Germany has advanced the price to about 10c. a pound at the present time. This certainly does not look attractive as a metallurgical proposition, but the possibilities of the cheap production of liquid SO_2 to permit of its being shipped considerable distances for use, are such that I think it will come to pass in this country in a comparatively short time.

E. A. CAPPELEN SMITH, New York, N. Y.—Our plant is practically finished; the leaching plant itself, as well as the mining plant, is practically completed. We have been slightly delayed with the completion of the power plant on account of the war in Europe—our power plant is being manufactured in Germany; according to our present plans, we expect to have the plant in operation by the first of April. So far in the construction work everything has gone on very nicely, and it now remains only to start the plant.

The general plan of work consists in steam-shovel mining the copper deposit followed by leaching the ore. From the crushing plant the ore will be moved by a belt and unloaded by a bridge that straddles the leaching tanks. The leaching tanks—six large concrete vats set end to

end—are each 160 ft. long, 110 ft. wide, and 16 ft. deep. Each tank is designed to hold 10,000 tons of ore. After the ore has been delivered to these by the traveling bridge from the main belt, the first solution is applied, by upward displacement. At first we had planned to conduct the operation entirely by downward percolation, but we found a slight tendency of the finer material to clog the filter bottom; also, we found that channels were formed in the charge. These difficulties were entirely overcome when we instituted the method of applying the liquor by upward displacement. The first treatment solution stands on the ore for 48 hr. This solution is immediately followed by the washing liquors in rotation, run on the top of the ore, and displacing the previous solution by the piston method. In this way, we find that we can wash the ore so that the tailing will contain only from 0.015 to 0.03 per cent. soluble copper. We think this is about the limit, and we can obtain this result without introducing any more water in the process than sufficient to make up for that carried off in the tailing as moisture, and in the solution which is discarded to get rid of the surplus acid produced from the ore.

The solution is removed from the tanks through lead-lined iron pipes, and conducted to a central pumping station, where we will use centrifugal lead-lined pumps, each with a capacity of 5,500 gal. per minute against a lifting height of 62 ft. The pumps have been built by the Worthington company; they have been tested in the manufacturers' shops, and as far as we can tell they are going to be highly satisfactory.

The solution is then delivered to the storage tanks, placed at an elevation above the leaching tanks, and from there it flows by gravity to the plant where we are going to eliminate the chlorine. We propose to pass the solution through cylinders 30 ft. long and 4 ft. in diameter, half filled with shot copper. By this means, the chlorine is precipitated in the form of cuprous chloride. The operation is continuous. The mixture of precipitated cuprous chloride, with the copper sulphate solution containing about 3 per cent. sulphuric acid, is run into Dorr thickeners, where the precipitate is separated from the solution. The cuprous chloride pulp corresponds to between 1 and 2 per cent. of the total amount of the solution. We are installing some Kelly filter presses, which are lead lined, and we are also going to install an Oliver filter, likewise lead lined, in order to compare the two types.

The filter-pressed material will be delivered to the smelter by the usual methods, and at the smelter it will be mixed with limestone and coke finely crushed (8- to 10-mesh material), put through a pug mill, pressed into briquets, and delivered to the blast furnace. In the blast furnace will be produced copper, and calcium chloride as a slag. The copper will be taken to a small reverberatory furnace, where it will be granulated and again used in the dechlorinators. We find that on account of the oxidizing agents present in the solution, partly in the form

of ferric sulphate and partly in the form of nitric acid, the efficiency of the copper shot is only about 50 per cent. of the theoretical; in other words, there is a direct solvent action on the metallic copper in addition to that caused by the reduction of the cuprous chloride. Practically all of the copper produced in the form of cuprous chloride will go back into the process.

The solution, after the chlorine is removed, is fed by gravity into the electrolytic tanks with a strength of approximately 5 per cent. copper. It will run through the depositing tanks, arranged in sections of 16 tanks each, which will have about 13,000 amperes to a tank. The voltage will be about 2.3. We expect a yield of approximately 1 lb. of copper per kilowatt-hour. The solution will leave the last tank containing about 1.5 per cent. copper. We find that the current efficiency remains high down to about 1.5 per cent. copper; if we go lower than that, the current efficiency will drop off rapidly. By holding to these limits we will have an ampere efficiency of approximately 90 per cent. The solution from the last tank in the cascade will be delivered back to the leaching plant to be used over again.

Our problem is somewhat unusual in this respect, that we have available sulphuric acid present in our mineral, and instead of having to look around for means of producing acid, we have to look for a method of getting rid of the surplus sulphuric acid. We will have to dispose of a quantity of the solution on account of its content of sulphuric acid.

There are no soluble impurities in the ore itself, such as antimony and arsenic. Soluble lime and alumina and alkali salts are negligible. Consequently, the usual trouble experienced in connection with leaching processes is not present in our problem. Chlorine, which we have in the upper part of the orebody, was really our most annoying difficulty, but the treatment of the solution with metallic copper has eliminated this difficulty. In addition to this, we also have a small amount of nitric acid in the form of nitrates. So far we have not been able to determine that any especial difficulty arises in the process itself from the presence of nitric acid. It has, however, a rather deleterious effect on lead connections, lead pipes, and lead anodes. We have noticed, however, that this action occurs only at points where we are handling the strong solutions—those containing 8, 9, or 10 per cent. sulphuric acid. Nitric acid is present up to 0.8 per cent., but outside of the difficulty mentioned, it does not give any trouble. Its presence does not seem to have any particular function in the process itself, and we have not been able to determine that its being there causes any lower current efficiency. It does not seem to have any effect on the voltage.

Chlorine seems to have a tendency, which was to be expected, of lumping the deposit to a certain extent, and the use of colloids to prevent this does not seem to be very encouraging. I think it will be a question,

more than anything else, of watching the deposit, and perhaps not allowing the cathode to get quite as thick as in ordinary practice.

Underlying the large oxidized orebody, at present developed to the extent of 200,000,000 tons, we have a sulphide orebody. Small-scale tests conducted on this material show that it can be treated either by roasting and leaching, or by water concentration followed by smelting in the usual manner. This orebody is also of enormous extent, but details have not as yet been decided upon for the final working of it, as the 200,000,000 tons of oxidized ore already developed will keep us busy for some time to come.

GEORGE D. VAN ARSDALE, New York, N. Y.—During the last few years we have been doing considerable experimenting on methods of precipitating copper. The first patent on precipitating from sulphate solutions by heating with SO_2 under pressure was taken out by us (U. S. Pat. 723,949).

This reaction, $\text{CuSO}_4 + \text{SO}_2 + 2\text{H}_2\text{O} = \text{Cu} + 2\text{H}_2\text{SO}_4$, is interesting, but there are a number of practical difficulties which will be met with in carrying it out on a scale of any size, the main ones being that it is not easy to get a sufficiently strong solution of SO_2 to carry out the reaction, and that the design of an apparatus for heating large amounts of corrosive solutions under pressure is a considerable problem. If these can be overcome, I believe the process may be developed, but I agree with Dr. Ricketts's opinion that electrolytic methods of precipitation are preferable.

Regarding our work at Douglas, we have felt that to say much about it was a little premature, as experiments are still in progress, but some notes on the preliminary experiments may be of some interest.

The general method which we adopted for the tests was sulphatizing roasting, leaching, and electrolysis from sulphate solutions, using depolarization.

The reason for the consideration of the sulphatizing roasting was that the materials which we have to treat contain in most cases large amounts of soluble alumina, and therefore, with a regenerative process only, we would be almost sure to have a deficiency of acid. With the sulphatizing roasting this deficiency of acid would be more or less made up.

The plant used for the preliminary experiments consisted of a 20-ft. six-hearth Wedge muffle furnace, several small leaching tanks, and an electrolytic installation designed to precipitate about 250 lb. of copper a day.

The objects of the roasting experiments were to determine the capacity of the furnace, the fuel consumption, and the percentage of extraction, both water soluble and acid soluble. The objects of the electrolytic experiments were to determine the power and the ampere efficiency, to determine whether the anodes which our preliminary experiments seemed

to indicate would be satisfactory under working conditions, and also to determine the best methods of dissolving and applying the SO_2 gas.

The results of our roasting experiments were as follows: We roasted three principal classes of material, in which the average insoluble copper in the calcines was 0.11, 0.16, and 0.41. The water-soluble copper showed 70 per cent. of the copper contents, 40 per cent., and 51 per cent., respectively, and the total extraction was 92.9, 88.0, and 70 per cent. One of the principal results of these experiments was to show us that for this class of roasting it was necessary to have more or less fine crushing as a preliminary.

The results of our preliminary electrolytic experiments were fairly satisfactory. We obtained a total amount of copper precipitated of something over 2 tons at an average figure for power of about 1.6 lb. of copper per kilowatt-hour. We produced acid per kilowatt-hour equivalent to about 5.1 lb. The ampere efficiency was low. This, however, was due to several reasons not directly connected with the process. We had a large number of short-circuits and other troubles, but we felt quite sure at the end of these preliminary experiments that we should get about 2 lb. of copper per kilowatt-hour. We had, however, a certain number of uncertainties and some anomalous results, so we started some further experiments on details of solution and other matters.

These further experiments resulted in the following general conclusions:

First, that it was not necessary to have such a large amount of SO_2 in solution as we had previously thought, and that, in addition, much better results could be obtained by using SO_2 together with iron, rather than either one alone. In other words, the function of the SO_2 was not only to act as a depolarizer, but also to act chemically in keeping the iron reduced. We also found there was a certain favorable effect from a high aluminum content. The temperature increase is favorable in the same way as in copper refining. With iron salts present there is a decreased ampere efficiency with increased temperature. This was counteracted by SO_2 and by increased current density.

As a result of these further experiments, we laid down the conditions of high ferrous sulphate in solution, high alumina, a temperature of about 115° to 130° F., current density of about 15 amperes per square inch, and sufficient sulphur dioxide present to reduce the voltage, and maintained sufficiently to keep the iron in the ferrous condition, and under these conditions I expect it will be possible to get a yield of from 2 to 2.5 lb. of copper per kilowatt-hour.

As to "foul solutions," which has heretofore seemed to be the principal bugbear in electrolysis, as long as the iron can be kept in the ferrous state, which can be done by SO_2 , it is fairly safe to say that, up to certain limits, better results will be obtained from so-called foul solutions than from pure copper sulphate solution.

We are now about to start a 75-ton plant, in which we expect to apply the information which we have gained to the treatment of one class of ore, and the result of the work which will be done in that plant will no doubt be published later.

RAYMOND F. BACON, Pittsburgh, Pa.—We have been in the field of copper investigation for some time, as many others have, but we feel it is somewhat premature to say very much as yet. We have been working on several different lines, and we believe no one leaching process will be developed that would be applicable to all ores or all conditions. There are certain leaching processes which are better adapted to the conditions of certain ores, and other processes are better adapted to the conditions of other ores.

As regards electrolytic processes, I think a good deal of the work we have been doing has been pretty well covered in different ways by the statements which have been made here to-night. Our work on electrolytic processes has been along two lines. First, to ascertain the optimum conditions for the electrolytic precipitation of copper from sulphate solutions obtained by the leaching of oxidized copper ores. In this phase of the work I may say that we finally obtained conditions under which we ran a solution over the ore through the precipitating tanks and then over the ore again, 20 cycles being completed, without any purification of the solution. There was precipitated on the average in these 20 cycles 1.86 lb. of copper per kilowatt-hour and the current efficiency on the twentieth cycle was fully as good as that on the first cycle. Our second line of work on electrolytic processes has been an attempt to find efficient and cheap depolarizers, so as to increase the amount of copper precipitated by a definite amount of electrical energy. We have found one very cheap substance which for short runs gave us 4 lb. copper per kilowatt-hour.

In regard to the sulphur dioxide process, we have been actuated by this thought—it is necessary in most ores to have quite an excess of acid to take care of mechanical losses and to take care of alkaline substances in the ore. One way of doing this is along the line which has been spoken of, by using the sulphur dioxide in an electrolytic cell. We, also, have used sulphur dioxide in an electrolytic cell, but all of our work has tended to show that the efficiency of sulphur dioxide as an anode depolarizer in a sulphate solution is rather low. Because of this low efficiency, our calculations showed us that it is cheaper to precipitate directly with the sulphur dioxide under pressure than to use the sulphur dioxide as an aid to electrical precipitation; that is, the heat required for direct precipitation with sulphur dioxide will be cheaper in most localities than the electrical energy required either for straight electrical precipitation or for the combination in which sulphur dioxide assists the current. We all appreciate the advantage which any electrolytic process has of giving directly cathodes

of electrolytic copper. On the other hand, it has seemed to us that for most localities and most conditions the disadvantages of electrolytic processes, as at present developed, outweigh this advantage. Some of the disadvantages of electrolytic processes are these: The fouling of solutions immediately and seriously cuts down the current efficiency. Changes in the character of the ore, particularly as regards soluble impurities, are always to be expected in large orebodies. The electrolytic process is very sensitive to changes thus brought about in the electrolyte. The installation and upkeep costs of the electrolytic precipitation process are very high and to obtain efficiency requires constant supervision by high-priced men. One rather fundamental objection to electrolytic processes has not been discussed here to-night. In electrolytic processes, at least of the type which have been spoken of to-night, the copper content of the electrolyte can be reduced in the precipitation cells no lower than 1.5 per cent. copper. That means that it is not easy to so conduct a cycle of leaching, precipitation, etc., that all of the water-soluble copper shall be removed from the ore and the volume of the solution in the cycle shall be kept constant. For example, let us assume in a systematic leaching process that all the copper has been extracted from a ton of ore and that that ore is then in contact with the solution which in the cycle is strongest in acid, having just come from the precipitation tanks, and which, of necessity, still contains 1.5 per cent. of copper. This one ton of ore is drained as thoroughly as possible and, according to our experience, then contains from 40 to 100 gal. of this solution containing 1.5 per cent. copper. In other words, there is from 6 to $13\frac{1}{2}$ lb. of copper still in this ore. By adding the proper amount of wash water to retain the volume of solution in the cycle constant—that is, in this particular case from 40 to 100 gal. of wash water—the copper content of that ore will be reduced to from 3 to about 7 lb. To wash out further this soluble copper, so as to have a negligible quantity of soluble copper remaining in the ore, according to our experience, requires an amount of wash water such as to increase very appreciably the total volume of solution which is in the cycle. It is obvious that in any process in which there is practically complete precipitation of the copper, as in Weidlein's sulphur dioxide process, this washing difficulty does not arise. In place of losing soluble copper in the ore, one there loses a corresponding amount of sulphuric acid; and in Weidlein's process sufficient sulphuric acid is constantly being formed, so that this loss can be taken care of. As to the element of danger in processes in which a solution is heated under pressure, I would call your attention to the fact that there are thousands of boilers operating continuously in the United States under pressures considerably in excess of those used in the Weidlein process. We built in Nevada a 40-ton plant and we have had certain mechanical difficulties, as we expected to have. These difficulties we are meeting by the installation of

a continuous process of precipitation. The advantages of continuous precipitation will be evident to all mill men. Our expectation that the cost of heat in this process of precipitating pure copper is less than the cost of electricity for electrical precipitation has been borne out. Of course, I recognize that there are localities where electrical power is so cheap that an electrolytic precipitation process should be the one to be considered, but I believe that for most localities in the western United States some of these other processes are more favorable as regards cost. In the Weidlein process, a very large excess of acid is constantly regenerated and all who have had experience in the leaching of ores know how desirable this is. In that connection I will say that we have developed a system of handling sulphur dioxide and a method of concentrating it from dilute flue gases. We believe we have made noteworthy progress in that direction, but I am not prepared at this time to go any further into that. In a short time we will be able to say a little more on the subject.

FREDERICK J. POPE, New Rochelle, N. Y.—The investigations in connection with the leaching of copper ores to which I will refer were carried on by A. W. Hahn and myself. In our preliminary work we were assisted by Messrs. Bryan and Aldrich of the Raritan Copper Works.

In our first considerations of the work, we had to decide: The solvent we would use; how we would precipitate the copper from the solution, and what we would do with the iron in the electrolyte. Without going into the details of the reasons, we decided that sulphuric acid would be the most satisfactory solvent and that we would recover the copper from the solutions by electro-deposition. As to how we would take care of the iron in the electrolyte, we had to choose between controlling the iron (*i.e.*, keeping it in the ferrous state) in the solution, and eliminating it from the solution. The removal of the iron appeared to offer the more attractive field.

Briefly, the method followed involves leaching with sulphuric acid, removing a portion of the iron from the solution, and precipitating the copper electrolytically.

Regarding the leaching *per se*, I do not think I could say anything which would be of any particular interest. There are certain factors which govern the extraction of copper from an ore, and when these factors are given due consideration, I think we can get about the same extraction with one process as with another.

In considering how to remove iron from the electrolyte, we were reminded of the work which had been done by Hoffman at Argentine, where he precipitated iron from bluestone solution. Hoffman accomplished this by heating the solution in a suitable tank, agitating with air, and adding copper in an oxidized form. I think he used a roasted lead-copper matte. This work of Hoffman's gave us our idea for the removal of iron from the solution.

For the copper oxide necessary in this work we have used high-grade copper carbonate ore, oxide formed by the roasting of cement copper, roasted high-grade sulphide ore, also roasted copper sulphide concentrates. In our demonstration plant (10 tons daily capacity of carbonate ores) we used roasted concentrates. The concentrates were roasted in a small Wedge furnace, the results obtained being far better than those attained in our preliminary work when we roasted the same class of concentrates in a laboratory muffle furnace.

In order to obtain a satisfactory reaction between the copper oxide and the iron, etc., in the solution, it is imperative to have the oxide material very finely divided. The roasted concentrates we used were ground in a small Abbé mill. We found that with 5 or 6 hr. grinding we obtained a product 93 per cent. of which would go through a 200-mesh screen. It is probable if we had had time to investigate this grinding operation, we would have found that it was not necessary to grind for 5 hr., but, owing to the shortness of the period at our disposal for securing results, we did not have the opportunity to bring this operation down to the shortest time limit.

From the Ajo carbonate ore we removed a little more than 2 lb. of iron per ton of ore treated. In order to keep the iron in the solution below the percentage deleterious to electrical efficiency, it was necessary to treat approximately one-quarter of the solution. We found that by manipulating the solution going on and off the ore in a certain way we could concentrate a large proportion of the iron in a certain part of the solution. This was accomplished as follows: The first solution going on the ore carried about 2.5 per cent. of free acid and was kept on the ore by circulating until it was approximately neutral. This neutralizing precipitated from the solution on the ore a considerable proportion of the iron, alumina, etc. This neutral solution was drawn from the ore, and after acidification by electrolyte coming off the electrolytic tanks was sent to electrolytic tanks for precipitation of the copper. As the first solution was withdrawn from the leach it was replaced by fresh solution. This fresh acid solution not only removed some more of the iron from the ore but also picked up the iron which had been precipitated from the first solution used. This second solution now high in iron was circulated until it was nearly neutral, when it was replaced by fresh solution carrying 4 to 4.5 per cent. free acid, which completed the leach. The total leaching period averaged about 80 hr. The solution high in iron, etc., was sent to that part of the plant where removal of iron and other impurities was accomplished.

In purifying the solution it was first heated to 195° to 200° F. When this temperature was attained agitation with air was begun and the finely ground oxide material was gradually added. Maintenance of a temperature of about 195° F. and agitation with air was continued for

3.5 hr. During the first hour all the ferric iron was precipitated. The removal of the ferrous iron was slower and we found that it was not advisable to endeavor to precipitate it all. In 3.5 hr. we easily precipitated 90 per cent. of the total iron and at the same time removed 65 to 75 per cent. of the alumina, 50 per cent. of the manganese, and all of the arsenic, antimony, and bismuth which might be present.

After the above operation the pulp was passed through a Schreiber wooden filter press. The solution came out beautifully clear and was sent to the high-acid electrolytic circulation system. We anticipated that with the precipitate produced we might have trouble with the filter cake; that it would be slimy and difficult to handle. We did not find it so. The cakes were 2 in. thick, firm, washed well, and the amount of copper in the form of unused oxide they contained was well within commercial limits.

As you are well aware, when the iron sulphate, aluminum sulphate, etc., react in the agitation tank with the copper oxide, they give up their acid radicals to the copper, forming copper sulphate, which in the electrolytic tanks yields sulphuric acid, so that the only loss of acid is that due to combination with the alkalies and that due to loss by entrainment in the tailing. This acid loss we found we could easily replace by a proper control of the roasting of the concentrates in the Wedge furnace; i.e., we could so control the roast as to maintain in the calcine sufficient copper sulphate either as water-soluble sulphate or as basic sulphate to compensate for the acid losses to which I have referred.

Regarding the electrolytic part of the work, we simply followed standard tank-house liberator-tank practice. Our anodes were antimonial lead, 4.0 per cent. antimony. We tried both grids and sheets. With the grids our voltage was 0.3 volt higher than with the sheet anode and we discarded the grids in favor of the sheets.

With reference to the use of a lead anode, I might say we commenced our work with some apprehension, fearing that there might be a serious loss of lead. We made a number of protracted tests and as a result we calculate that the cost for lead anodes at Ajo with lead at 8c. per pound would be less than 0.2c. per ton of ore treated.

In the process I have outlined there is really nothing new or novel. It simply consists of an idea picked up here and another there, and the putting of these various ideas together.

If we were to criticise the process, the pros and cons might be summed up about as follows:

Advantages:

- a. Simplicity.
- b. Every step in the process has been in use in commercial plants for years.

- c. Saturated solutions avoided.
- d. Wash water not excessive, and if it should become so, it can be taken care of.
- e. Impurities which would injure the quality of the electrolytic copper are eliminated from the electrolyte.
- f. Acid control very simple.
- g. No step in the process is dependent upon fine adjustments.

Disadvantages:

- a. High and low acids used. This necessitates more solution-storage tanks, more solution lines, and more pumps than if a single solution were used.
- b. There is an extra operation: namely, purification of electrolyte, which involves the installation of a roasting furnace, boilers, compressor, and filter press.
- c. The lead anodes will probably have to be changed at the end of four or five months, since the lead peroxide scale formed will increase the voltage.

J. PARKE CHANNING, New York, N. Y.—The problem presented in the Miami district is a little different from that in the districts referred to in the discussion of this evening, inasmuch as the ore to be treated there is one which is a mixed chalcocite and carbonate. The gangue, as you know, is particularly free from any impurities that go into solution, but the fact that both chalcocite and carbonate are present makes it a rather difficult problem. The particular ore deposit upon which we worked was that of the New Keystone, but the same class of ore existed in the Miami, and also on the Inspiration. The problem therefore presented to us was either a straight leaching one, which would have to be proceeded with by complete dead roasting, or else it would be a mixed process, water concentration followed by leaching, or possibly leaching followed by water concentration; and then finally, if flotation came in, we might have to make three bites of the cherry—that is, use water concentration, flotation, and leaching. You can readily see there would be a great many combinations; and whether it would be desirable to leach first, then water concentrate, and then float, or whether it might be desirable to make the sequence different, we do not know. This Keystone ore averages about $2\frac{1}{4}$ per cent. copper, of which anywhere from 30 to 40 per cent. is oxidized. After making a number of concentration tests we decided upon attempting a dead roast; in other words, converting all to the copper into a soluble form and leaching it.

It is not possible for me to give you the details of the numerous different things we tried. I may say, however, that for the purpose of precipitation, we tried the electrolytic method. The problem with respect to these mixed ores is by no means solved and we are still conduct-

ing experiments along these lines, and I think Mr. Canby, the gentleman in charge of the experiments, can give you in a general way some idea of what we have accomplished and what is still left undone.

R. C. CANBY, Wallingford, Conn.—The only thing I can think of which has not been mentioned is the circulation of the electrolyte. In our electrolytic work I designed the cells so that the solution, instead of flowing lengthwise against the flat surfaces of the anodes and cathodes, flowed crosswise of the cell. In that way, in the use of sulphurous acid gas, I had hoped, by having a rapid circulation, to use a much smaller quantity of both copper and the depolarizing element in the electrolyte, with a correspondingly stronger current, and also to be able to precipitate the copper without heating the electrolyte. I was able to use a current density as high as 10 or 12 amperes per square foot with the temperature of the electrolyte about 18° to 20° C., and get a very satisfactory coherent deposit. I think that was the only novelty in the electrolytic work.

The other things were done very much along the lines which have been discussed here this evening. We think, however, it would be very much better if all of the oxides could be treated in the present plant, and we are now working to that end, so as not to have a separate installation for the semi-oxidized ores. It is along that line that experiments are now being carried on, so as to treat the oxidized material, which is mostly in the flotation tailings. I think there were perhaps five or six different methods of manipulation tried, such as leaching resulting slimes and leaching before crushing for concentration, etc., but all of these combinations contained really nothing of any special interest, as they are thoroughly familiar to everybody.

J. PARKE CHANNING.—How about roasting with the oil furnace—would that be of interest?

R. C. CANBY.—That might be of interest. The problem was to use our 14° Baumé fuel oil, to be burned at a temperature of 2,000°, and yet keep our ore bed at not to exceed 800° to 1,000° C. I accomplished this by using a furnace similar to that used for the Huntington and Heberlein process at El Paso, and at three other American smelting plants, the hearth revolving past the flame, so that I was able to roast the ore without at any time bringing the temperature of the roasting ore above the desired 1,000° C., whereas the fuel was being burned at about 2,000°, or a little over. The furnace was entirely of reinforced concrete construction and stood well.

RICHARD LAMB, New York, N. Y.—During 1907 I designed and erected a leaching and electrolytic copper extraction plant at High Hill, Va., on the Virginia Copper Co.'s property, to treat the bornite and chalcocite ores found there.

The average ore analyzed about as follows:

	Per Cent.
Copper.....	3.0
Sulphur.....	1.5
Silica.....	89.0
Lime.....	0.75
Iron.....	1.5
Magnesia.....	3.0
Alumina.....	0.35

Owing to the large percentage of silica this ore is difficult to concentrate economically. The large bulk of silica crowds off the values on any form of concentrating apparatus, and at least 50 per cent. of the values is lost if machines are worked at a speed necessary to make them efficient. There is a large body of copper ore in the Virginia copper belt, but many failures have been made in the attempt to concentrate it. However, the ore is ideal for leaching. The magnesia and alumina are mostly in silicate form, and do not go into the acid solution, and the large percentage of silica is in its favor when leaching.

I built the first experimental plant for treating this ore in Hoboken, and first tested leaching the ore in open tanks. In crushing this ore to say 10 mesh at least 25 per cent. goes into slime. This adds to the difficulty of leaching if the usual leaching tanks are used. I therefore substituted a chlorination barrel for the leaching tank in order that the crushed ore might be stirred while leaching. Stirring arms in open leaching tanks when using sulphuric acid disintegrate quickly. The lead-lined barrel used stirs the ore without injury from the acid. As the ore is a sulphide it has to be roasted. In the Hoboken test plant I used a small Brückner roaster. In the large test plant at the Virginia Copper Co.'s mine I used an Edwards roaster with a capacity of 30 tons of ore per 24 hr. I was able with this roaster to control the heat. While we had only 1.5 per cent. of iron in the ore, to avoid its polluting the electrolyte I sought to change it from an oxide to a sesquioxide of iron, which is insoluble in dilute sulphuric acid, by keeping the heat while roasting to about 900° F. Dr. James Douglas told me that he had succeeded in changing the iron to a sesquioxide of iron in a copper ore he was leaching, having over 25 per cent. of iron in same, by roasting with the proper degree of heat. I found that if the ore is crushed only to a size that exposes the sulphide on the surface, the roast will be as good as it would be if the ore was ground exceedingly fine. There is some oxide and carbonate of copper ore in the form of cuprite, malachite, melaconite, and green carbonate produced, but these are not separated when coming out of the mine, so they were sent to the roaster, although the roasting was superfluous, as sulphurous acid will take the copper of those oxide and carbonate ores into solution without their being roasted. The ideal

roast would be to change the sulphide ores into sulphate, but since this condition would require careful manipulating I sought to make the sulphide ore oxide and sulphate, but not to make a sweet roast, only seeking to leave as little of the ore in a sulphide condition as possible. If the ore could all be roasted to a sulphate condition the sulphur of the ore would go into the solution while leaching and thus add sulphur to the acid supply.

The ore, ground to 10 mesh and roasted, was put into a lead-lined barrel. A solution of sulphurous acid was made by burning sulphur in an inclosed oven with a supply of air sufficient in volume to supply the oxygen to produce sulphurous gas, and under a pressure just sufficient to overcome the hydraulic head due to the height of the water from the submerged outlet of the sulphurous gas delivery pipe to the surface of the water in the tank. The water is acidulated to about 10° Baumé by absorbing the sulphurous gas.

This acid is pumped into the leaching barrel with the roasted ore, and the barrel is revolved until the copper has gone into solution. This takes about 3 hr. Compressed air is then passed into the barrel and the solvent is forced from the barrel through the filter in same. This filter is the same as is used in the barrels used in chlorination work. This solution entered a 20,000-gal. cypress stock tank, painted with Mogul acid-proof paint. The elevation of the tank was such that gravity would produce a flow of the solution or electrolyte from the stock tank through 16 lead-lined cypress vats, placed at elevations with reference to each other so as to continue the flow of the electrolyte through the 16 vats. An acid-proof pipe line was also provided, situated below the vats, so that any vat could be discharged or shunted. Each tank was provided with $\frac{1}{8}$ -in. thick lead anodes and cathodes, with exposed surface 25 by 36 in., in all 1,030, the cathodes providing over 6,000 sq. ft. of depositing surface. An overhead crane was provided to carry the anodes and cathodes to their respective places.

The electric current was supplied by two Holtzer-Cabot 30-volt, 1,200-ampere generators, with a 120-volt, 1.5-kw. exciter.

By the use of sulphurous acid the anodes were prevented from peroxidizing. The sulphurous acid of this electrolyte is changed to sulphuric acid by the electrolysis and becomes a stable acid. After passing through the vats any loss of acid from being neutralized by the alkali of the ore is replaced by sulphurous acid, as before described, and the electrolyte is used over and over again to leach more ore. When the electrolyte becomes polluted it is run into a larger vat having an extra number of electrodes so that the current density is very slight and the copper is practically depleted. The electrolyte from this hospital tank can be run to waste or further treated for its sulphur contents.

The voltage supplied to each tank was 1.8, and a current density of

3 amperes per square foot was used. The electrolyte was permitted to run from $\frac{1}{4}$ to $\frac{1}{2}$ per cent. in copper, when used over for further leaching.

The output was 30.6 lb. of pure electrolytic copper per horsepower-day of 24 hr. With water power at \$12 per horsepower per year this would represent a cost of 0.12c. per pound for electrolytic treatment.

With steam power costing \$48 per horsepower per year the cost for electrolytic treatment would be about $\frac{1}{2}$ c. per pound of copper produced. Producing copper in the Virginia copper belt, from underground to finished product, would cost about 6.9c. per pound working upon a scale of say 5 tons of copper per day. In my belief, using the large available water power nearby, and making 5,000 tons per year, the cost of copper will not exceed 6c. per pound. The capacity of the trial plant described was 2,800 lb. per 24 hr. In building this plant I was employed by one man, who alone, on the advice of his brother, an experienced metallurgist, undertook to lease the Virginia Copper Co.'s mine and put up the experimental plant. The plant was run only a few days, the owner having lost his money in the fall of 1907, and the plant was thrown into the courts, with disastrous results to the owner. Enough was done to demonstrate that with the above described combination of well-known and tried-out apparatus, using the cheapest of solvents, sulphurous acid, there is no reason why a suitable copper ore should not be treated on a large scale so as to produce electrolytic copper at a lower cost than by the old dry-process methods.



Pyritic Smelting

A Discussion at a meeting of the New York Section, Dec. 2, 1914

D. H. BROWNE.—If 35 years ago we had met to discuss the subject that is before us to-night, the criticism that we must all be mad or we would not be here at all would have been perfectly applicable. No group of men would have come together a generation ago to discuss the subject of smelting ores without extraneous fuel, or largely by the oxidation of their own impurities, without incurring the imputation of being crazy.

But metallurgy, like all other branches of human industry, progresses by idealism. In every generation there are old men who see visions and young men who dream dreams.

Thirty-five years ago John Hollway conceived the idea that in the Rio Tinto ore the sulphur and iron, if properly oxidized, could be made to produce heat enough to carry on the smelting of the ores. He saw the vision. Twenty-five years ago Mr. Austin, of Montana, took out a patent on a furnace in which to carry out Hollway's ideas. He dreamed the dream.

Twenty years ago a man working in Montana grasped the possibilities of this idea. To him has been given that inexpressible pleasure which comes from realizing the idea and making the concept concrete. Gentlemen, it gives me great pleasure to introduce to you the man who made the ideal real, Robert Sticht, of Tasmania.

ROBERT C. STICHT, Queenstown, Tasmania.—I find it extremely difficult to act up, in the smallest degree, to the encomium passed upon me by Mr. Browne.

It is true it has been my good fortune to be able to put some sort of method into the "madness" of 25 years ago; but I am not the originator of the idea, nor the only one who has carried it out. Mr. Browne mentioned the originator, who undoubtedly is Mr. Hollway. He also mentioned Mr. Austin, who gave the first instigation to pyrite smelting as now practiced. As against both of these men I was fortunate enough to be able to command the necessary natural resources to carry out the idea continuously in a practical way, so as to make it permanently possible.

At the time I left for Tasmania the method had been tried in this country, and it was my good luck to be connected with some of those efforts, and also to become posted on what Mr. Hollway had attempted. Fortified by his reasonings I was able to apply them to the blast-furnace treatment of pyrites, which he had not the opportunity of carrying to

success, and to experiment with this method of treatment in two States, Montana and Colorado, in a small way, now and then.

I say "now and then," because, as a rule, the primary conditions were not favorable, and with this process, as with so many other "processes," it is much more difficult to get an ore which suits the "process" than a process which suits the ore. As remarked, the occasions for trying the real thing were few and lasted but a short time each.

Nevertheless, I thus had a certain foreshadowing of what course would ultimately lead to permanent success, gathered at Toston and Boulder, Mont., and Kokomo and Leadville, Colo. So when I got to Mount Lyell I had some idea how to proceed. I also had perhaps the necessary American "cheek" to go there on so small a basis, but I had every reason to believe that the conditions of ore supply, etc., in Tasmania were such as to make this new method particularly applicable there. I knew very little about the inner workings of pyritic smelting, but was one of three or four men in this country who knew how to do it practically—i.e., empirically.

As time went on, my ideas became concrete, and now we feel we know its real scope and limitations. We know that in some territories it has been tried and abandoned, and in some cases I think more might have been done with it had metallurgical success not been interfered with by economic factors, the mere question of profitableness many times bringing the carrying out of an interesting theoretical idea to an untimely end. How well we all know this!

I understand questions will be asked during the course of this discussion, which has been honored with the title of an "address." I have not written any address, as I am on a vacation trip and do not intend to do any hard work. I have no figures at hand, and I think the general spirit of this meeting makes it unnecessary for me to produce them. It should resolve itself into a conversational discussion of the subject.

To begin: Of course you all know where Tasmania is. It is that little red spot on the map, among the possessions belonging to England, which occupies a southernmost position, immediately below the continent of Australia, and, except in the north, is surrounded by a tremendous amount of ocean in every direction. It is a small island with extensive mineral resources. Generally speaking, they are poor in values—low-grade propositions; but tin, lead, copper, silver and gold, and rarer metals and earths come from there.

On the west coast of that island there is a low but rugged mountain range, and in the center of that there is the orebody to which my attention was called.

I went there in 1895, single-handed; and if there is anything with which I am pleased it is the fact that I went with the knowledge I had picked up, and was given a free hand to put my modicum of experience to the test. Had I not been an American engineer I would not have been

given the free scope accorded to me; and, consequently, what I have accomplished in Australia is due to the high regard which the American engineer enjoys all over the world. All the outlying places of the earth show the work of our hands, to the honor of our profession.

In Australia one American metallurgist had already made a signal success. It was H. H. Schlapp, of Broken Hill fame. We knew each other, and his belief in my statements, which to many must have seemed bordering on the impossible, was of service in giving me support. Dr. E. D. Peters had written a report on the property and gone into the question of treatment, but, owing to the novelty of pyritic smelting, both he and Schlapp were naturally inclined to go slow at first and start with the ordinary process. One day, however, I was asked at a directors' meeting, what would I do if the mine were my own. I said, "I would start with pyritic smelting right away!" and the Chairman clinched the situation by responding, "Then go ahead your way!"

We started in a wilderness, and got a plant finished, with three furnaces and hot-blast paraphernalia, in a year's time. The transportation of the machinery was very difficult, as there was no railway at first. We had to build that simultaneously and it was completed just in time to settle the transportation difficulty. The latter did not allow us to get in over 150 tons of coke per month, and I knew I could use that up in one week if pyritic smelting would not work.

As remarked, we erected three furnaces. We started the first one June 24, 1896, and gradually enlarged the plant up to six furnaces. Then, not being able to smelt more than 500 tons a day in four out of those six furnaces, we put up a second plant with five more furnaces, so as to smelt 1,000 long tons a day. Nowadays 1,000 tons a day is not much, but this tonnage was a kind of standard at that time, and has so remained in Australia longer than in this country. Tonnages are, of course, on the increase everywhere, but, coming from foreign parts, when one visits your plants and sees 12,000 tons of ore smelted daily in a single one of them, one is simply overwhelmed, and proud of the colleagues who are doing it.

We soon became an important producer of an eminently clean copper. For a number of years it was thought necessary to smelt the ore twice, first into a lower-grade matte and then into converter matte. It was some years, I think something like six years, before we discarded this system of double smelting, and made converter matte out of the ore in one operation. The re-smelting of the matte, so as to enrich it, was an early trick of the pyritic smelters in the United States, to reduce freight and treatment charges on the matte, and was done by them at once. They found the matte much easier to smelt than the ore, and practiced this concentration at a time when the big smelters still thought it could not be done.

We also installed hot blast, and used it constantly on both ore and matte. In fact, in the first few years we could not have smelted properly without it, but finally the discovery that the amount of energy communicated to the reactions in the furnace by that hot blast was very little led to the conclusion that it could be abandoned without fear, and the turning point came when I suggested doubling the amount of blast per furnace by putting on two blowers, in place of one originally provided for each furnace, and using it cold. This brought the work a step nearer to Bessemerizing, which has always been my ideal point of view for the explanation and evaluation of anything connected with pyritic smelting. The pressure went up, so did the grade of the matte, and costs went down considerably. We then had 11 furnaces, and we discarded one plant altogether, once we found that, with an increased amount of air per furnace, we could get the same capacity out of one plant, and we cut the works down to one set of furnaces, five at that time, but now six.

Since then (1902-1903) we have been going the even tenor of our way with but very little metallurgical excitement, though constantly improving both plant and method. I am sorry to say, however, that our most ideal days are behind us. We now use more carbonaceous fuel than we used to and also treat not quite as much pyrites as before—say 850 tons a day, and 450 tons of siliceous ore, making 1,200 to 1,300 long tons a day in between two and three furnaces. We currently blow about 20,000 cu. ft. of air a minute into a furnace. The pressure is great—64 oz., sometimes 65 and 66. This is largely due to the fact that we have progressively increased the height of our column, so that now it is at times nearly 18 ft. high, above the tuyères.

MR. GUESS.—How long are those furnaces?

MR. STICHT.—They are 210 in. long and 54 in. wide. We are not in the happy position of being able to utilize a furnace which is very long, as we practically smelt from hand to mouth, and, for instance, would be embarrassed for ore supply over the week ends, and at times of very boisterous weather, when the mine output drops.

The capacity of the works is greater than the capacity of the mines. The latter is decided by special considerations, and a very particular one is the value set for the siliceous ore we mine, and the extent to which it is extracted. There are great low-grade reserves in this mine, but not profitable at present reckoning, otherwise we would be working 2,000 or 3,000 tons of ore a day.

The orebodies are not large enough for such great undertakings as you have here. I am sorry I cannot tell you about them, as the deposits are most interesting.

I mentioned the siliceous ore. We started without it, and, for a number of years, used barren quartzite and sandstone devoid of all metal values, for a siliceous flux. We had a pure pyrites from the start, free

from all deleterious substances, and the gangue matter amounted to 8 per cent. at the time. It has gone up until now it is 12 or 14 per cent. The quartz we used had no pyrites or other value in it. Although gold is present in some of the siliceous rocks there, we could not use them on account of excessive cost for delivery.

Early in 1903, a neighboring company, known as the North Mount Lyell Co., which had no connection with us, originally, failed. They had been fortunate in the development of a property which was at first hardly more than a wildcat, had erected smelters and built railways, both in imitation of and in opposition to ourselves; and finally had to give up work, after practically spending £800,000 for nothing. In the meantime they had sold us 80,000 tons of their ore, so that we were fully familiar with it. When the moment of their reckoning came they approached us with a proposition to amalgamate, and this was eventually done by the two companies simply coalescing. The works they had erected became superfluous and their mine was taken over by us, their various staffs being paid off. The failure of the North Lyell company was due primarily to professional incapacity, *i.e.*, inability to see that, under the local conditions, it could not possibly pay to treat a highly siliceous copper ore, of even 10 per cent. copper, and practically devoid of silver and gold values, by ordinary matte smelting, either in blast furnaces or reverberatories, both of which they tried. Nature had meant the two mines to be worked together as they are now being worked, and nature finally had her way. Our company had the stronger hold on nature through possessing the pyrites, which the other company practically lacked. For our part, however, we were willing to amalgamate for the reason that the copper value in our pyrites had very much decreased and was now, for a large part of the body, only $1\frac{1}{2}$ per cent. We had gradually come down from an initial $4\frac{1}{2}$ per cent., through $3\frac{1}{2}$ per cent., to $2\frac{1}{2}$ per cent., or rather 2.35 per cent., which we maintained for some years, and this was also getting scarce. It had always been known that an important portion of the deposit was only $\frac{1}{2}$ per cent. in copper, but this had been regarded as beyond the pale of treatment, though now it is the best we have. The accession of the North Mount Lyell mine came at an opportune moment for both parties, for we have since established its value beyond the dreams of its promoters, and have paid the North Lyell shareholders dividends which they would not have obtained otherwise.

And now it must be remembered that we work these two mutually complementary ores, and only two, that those ores are very pure, that the siliceous ore has free silica in it up to 70 per cent.; that our pyrites is practically free from zinc and lead, and that there is not too much alumina or heavy spar, and that from the metallurgical point of view all conditions are very favorable. Of course we treat small quantities of less easy ores, but they do not affect the average.

We have plenty of water, and quartz, clay, and limestone handy, and a highly specialized staff. So the Mount Lyell company has made its reputation on the strength of those two ores. We have not had any experience in smelting bad ores, or middling bad ores, such as the process has been tried on in this country. We have never had to solve certain problems with which you have been confronted. The only material which is difficult there is middle-class siliceous ores with 45 to 50 per cent. of silica, combined, which are really hydro-mica schists, impregnated with copper pyrites. These run too low in copper value for us, and we cannot do anything profitable with them until the flotation processes are perfected.

It will interest you to know that the superintendence of the smelting operations has always been in the hands of Americans, George F. Beardsley at first, after myself; then A. Lewis Dean, and now Robert P. Roberts. The other officers are all Australians, and a most excellent lot of men also. As is but natural under the circumstances, owing to isolation, etc., the personnel of the staff has not changed greatly since the beginning, and that is true in all ranks of service.

Circumstances have been favorable to our consistently smelting about a 2.25 per cent. copper ore. As remarked, the Mount Lyell pyrites ran that after a short period of higher assay, and now that it has come down to 0.5 per cent., or even only 0.2 per cent., it is being treated together with the North Lyell ore, which as mined assays 6 per cent. We thus still have a 2.15 to 2.25 per cent. average and out of this we produce a 45 to 50 per cent. copper matte, by one smelting, at the rate of over 1,000 tons of ore daily, in an average of a little over two furnaces. Unfortunately for the maintenance of the purely pyrite state of the smelting, *i.e.*, a nearly cokeless smelting, the Mount Lyell ore has got lower in iron and sulphur by a few per cent., and the further fact that its copper value has fallen so considerably makes us give the preference to the North Lyell ore as much as possible. The effect has been that we now use a great deal more coke than we used to. Instead of having 42 to 44 per cent. of iron the Mount Lyell now has only 35 per cent., and the corresponding sulphur, and the conditions are such that one cannot operate with the low coke percentage of the older practice.

At one time the amount of coke on charge went as low as 0.1 per cent. per half year. Now, we use from 3.5 to 5 per cent. and occasionally we exceed that a little. That is simply due to the fact that conditions have altered. The composition of the slag has altered too, though it still forms itself, as before. Instead of running 30 to 32 per cent. in silica, as it used to do in the past, it now goes 36 and 38 per cent. The iron in the slag, instead of being 52 per cent. FeO, is now 45 per cent. FeO. We have always used some lime, and the amount on charge is kept constant, irrespective of the consequent slight variation in the CaO contents of the

slag. The alumina is not very abundant, about 7 per cent. in the slag. The copper content of the slag from this high concentration smelting, of 20 into 1, may be a little higher than it would be in the double smelting process, but we have not assured ourselves that it really is.

We have about the usual relation between the copper contents of the slag and the matte, and the average slag assay is from 0.35 to 0.45 per cent. copper, according to the grade of the matte. Our slag stream runs 50 to 60 ft. over forehearth before it is discarded into the drain, where it is washed away, and we catch a little matte in each compartment.

We have never refined our own copper, for the reason that the disposal from Australia was difficult and relatively unprofitable. For many years the blister went to Baltimore, and several of the important questions affecting the sampling of this material were pioneered on our product. The situation at the present time is that since the Mount Morgan company's property in Queensland has developed from a gold mine into a copper mine, and circumstances have obliged them to go into refining on their own account, they have given us better prices than any one in America, so it is not only patriotism, but also business, to send the material to them. Just at present, although the European war is on, our company is running with its accustomed regularity and on full capacity, and the Board thinks we can continue as long as the war may last. The copper goes to England now, whereas before most of it went to France and Germany.

Our labor conditions are unique. You hear much of how bad things are in Australia in that line. Certainly, in some respects, they could not be worse in connection with certain classes of men, such as the wharf laborers, and there are constant strikes in most ranks, but that must be put down to a certain hysteria which is being engendered in the men by the new-born consciousness that they are now playing a part in the affairs of the world at large which was not vouchsafed to their forebears. In Australia, I think, this condition of neurosis will pass away early. As you are aware, the relations between capital and labor are there regulated by some very wise and prudent laws. The principal drawback is that the particular judge of the Supreme Court who handles the arbitration cases is too much of a pure lawyer, and not enough conversant with the peculiarities of the human material he deals with, for his decisions to be wholly satisfactory. We have had very little trouble with our own men. We had one strike over a matter of discipline, but the men went back to work unconditionally as far as this question was concerned, after three months of cessation of work, and we gave them higher wages afterward, without their asking for it.

Outside of us the mainland companies have frequent labor troubles, though much of this is due rather to psychological than to economic causes. As far as Mount Lyell is concerned it must be borne in mind that we are

decidedly a "one-mine camp." In addition to the parent mine (Mount Lyell) we also own the North Lyell mine and practically all the other properties in the Lyell district. The importance of our enterprise to the State of Tasmania may be gathered when I say that it is generally held that without us the State would go bankrupt. We have one-third of the mine employees in the State in our employ, and are regarded as a patriarchal kind of concern. Our position makes it both necessary and possible for us to do things not feasible under more circumscribed conditions.

In the case of the fire of October, 1912, in the North Lyell mine, we ran the smelters for over six months with almost no profit for no reason but to keep the men together. It would have been highly imprudent to shut down, as this would have depopulated the district, and the resumption of work would have given us endless trouble and expensive delay.

Relations with the unions are quite amicable, and just as we have not had any very serious trouble with them so far, I do not anticipate any in the future. They are legally responsible the same as the employer, and want it so. This must be remembered, that in those countries all classes are exceedingly law-abiding; this applies to the lowly working man as well as to the highly capitalistic citizen. There is also a reasonable conservative element among the men, which keeps them in check. The word "socialism" is wrongly applied to many things going on in Australia. The men at heart are simple, decent, fair-minded men, and not adherents of the red flag. They like to have property of their own. There are only 4,000,000 inhabitants of all classes in all Australia, and 1,000,000 in New Zealand, so the population is widely acquainted within itself, like the units of a big city, and it is a fact that they can enact public measures there which could not yet be enacted here. Life is less complex than in the United States, and has an old-world ring about it even in the mining camps. Of course the men have their peculiarities, which are inclined to both amaze and irritate an American, but once understood they are agreeable to handle and most loyal. The nationality of the men is practically all British. Legislation totally forbids the entry of the Asiatic races and of the southeastern European peoples you have here in such masses. There are almost no Italians and but a few Greeks, and not even very many Irish. On the other hand, there are very many Scotch. The Cornishmen, once prominent there, are less so now, and there are no Welshmen of importance anywhere in the smelting line. Scandinavians and Germans are scarce.

I now seem to have traversed a good deal of ground without giving much information. Will not some one kindly ask a question?

MR. WIERUM.—Could you tell us the reason why the percentage of coke has gone from 2 per cent. to 3 or 4 per cent.?

MR. STICHT.—Expressed in simplest terms, we now have to use the extra coke to keep the slag hot enough. It simply seems to add a certain

amount of heat at a point far above the tuyères, in the preparatory region, which is necessary to make it feasible to do the smelting with satisfaction. We cannot get on with $1\frac{1}{2}$ per cent. of coke on the charge. As intimated above, the fuel values of our pyrites in iron and sulphur have fallen off. Then again, we wish to conserve the pyrites and smelt relatively more of the siliceous ore. This does not necessarily mean that the slags must show more silica in percentage. Of course the additional coke has this effect, but a simultaneous higher degree of oxidation of the iron would counterbalance this and bring the slag composition back to the old figures. As we oxidize 95 or 96 per cent. of the iron we feel we go far enough in that connection, particularly as we get a 45 per cent. matte at the same time, and do not want a higher tenor, consequently our slag now falls more siliceous than it used to.

The coke we use is made by our own works at Port Kembla, in New South Wales, and arrives with at least 10 per cent. water in it, and has 16 per cent. ash. It is purposely made as hard and dense as a brick, and is discharged from the ovens in huge pieces. It is carried in bags, and breaks up a good deal on the long journey, though always lumpy. As regards the action of the coke, it is my opinion that unless a great deal of it is used, it does not even get down to the focus, but is consumed above that place, furnishing a modicum of heat only, by reaction with the SO_2 . As regards the preferential smelting of the siliceous ore, one cannot go very far in this. We started with two parts pyrites to one of siliceous ore and kept this up for years. Now we treat 1.3, 1.5, 1.6, and 1.7 of pyrites to one of siliceous ore. We try to smelt one to one, but that is out of the question. To do that we would have to depart greatly from the present standard into the region of expensive smelting with the use of lime in quantity, likewise coke.

The decreased iron and sulphur contents of the pyritic ore is due to the coming in of small amounts of galena and blende, and a slight rise in the silica, etc., with depth in the mine.

E. A. C. SMITH.—I would like to ask about the moisture. I understand you have heavy rains. As the moisture in the ore changes would not that affect you?

MR. STICHT.—The moisture in the ores is pretty constant and hardly goes over 5 per cent., usually 2 or 3 in the ores, and in the limestone too. All these materials are dense and not earthy. The moisture is merely superficial and, as it is always raining, it is always the same.

MR. LLOYD.—In the matter of detail of furnace operation, when you were using a maximum charge of coke, was it so much charge and so much coke? In other words, was there anything peculiar in the general feeding of the furnaces? Is the coke put in charges, or is it put in here and there as the feeder may determine?

MR. STICHT.—No. The coke is pushed over the edge of the charging

plate evenly for the length of the furnace, first, and the ore follows it. We do not put the coke in distributed over the whole surface of the charge, and seldom use extra coke charges. We try to get it in on the long walls of the furnace just as if it got down before the tuyère, but it never really gets there, being consumed far above.

The very low coke percentages were achieved during the hot-blast régime, and we got down to 0.1 per cent. We then used one ton of pyrites, with sometimes as little as 500 lb. of quartz, or else concentrated first matte, so we had a superabundance of iron and sulphur as compared with the present.

The pyrites is the standard—that is, it is always the same in weight on charge. The coke varies, but we vary the silica much more. I may explain that part of the coke is put in merely to allow the metallurgist and his assistants to go home and sleep. The metallurgist and his crew are only human beings, and we spend a lot of money annually keeping them sweet and fresh, and in good vigorous and healthy running order. If every one round about the furnaces was a highly qualified academic chemist, who could give his unremitting personal attention to the descent, within the furnace, of every piece of ore and flux, then we might, perhaps, permanently omit the coke altogether. But ordinary flesh and bone cannot do it. We have, of course, occasionally smelted without coke for a day or two, but cannot keep it up.

E. B. KIRBY.—What is apt to go wrong when something does go wrong?

MR. STICHT.—Whenever we stop a furnace, it is because of the forehearth, or of water getting inside of the furnace from leaky jackets. When things go wrong it is not due to any form of sows. Our furnace campaigns used to be short, but of late years it has been easy to keep a furnace going for six months and more, even with leaky jackets. There was at times an overdose of silica, which could not scorify. That is, so much quartz in excess of that really needed was fed in that the surplus was left behind, and after a while it choked the furnace and it stopped. We called that a “silica sow,” but it is not a true sow.

I may explain that we avoid anything that chokes the furnace and put the sulphide and the limestone in as big as a horse's head—anything a man can lift, or that can go through an ore gate. Sometimes a long piece goes through endways, but we do not usually smash it on the charging plate unless it is schisty ore.

KARL EILERS.—Did you do that when you were at your old furnaces?

MR. STICHT.—Yes. We especially try to keep out all barren, valueless schist which may find its way into the bins, as it contains only 35 to 45 per cent. silica, combined with alumina.

MR. EILERS.—What would happen if you had to smelt only such bad stuff?

MR. STICHT.—We would give it up, I think.

BRADLEY STOUGHTON.—You spoke of it not being possible to make furnaces go higher than 18 ft. Is it not possible to make them in copper smelting?

MR. STICHT.—Yes. I have counseled their erection under certain circumstances, up to 25-ft. columns, though this may be excessive. It is a question to be decided between the sticking quality of the ore charge and the blowing machinery. With little blast too high a column leads to scaffolding. The introduction of turbo blowers allowed us to increase our column, and we have been greatly satisfied with them since. We are now putting in water power, and we have arranged for facilities to increase the pressure up to 8 lb. by means of multistage air compressors of this type. In the matter of furnace column I feel we have not yet reached finality even for our conditions.

MR. LLOYD.—The condition of the interior of a furnace running pyritically is different, widely different, from that of a furnace working under fuel, is it not?

MR. STICHT.—The subject of the descent of the masses in ordinary work in the furnace shaft is fully dealt with in the text books and the conclusions reached may also be confirmed by experiment in glass models. But in the pyritic furnace conditions are much more complex and impossible of such visual demonstration. In our own case it is decidedly so. I do not know how the inside of the furnace looks when working, but I imagine that a very important part is played by the silica in lumps so large (about 6-in. chunks) that when they have decrepitated they do not block the furnace but leave open crevices between the separate pieces. This porous silica bed rests on the bottom of the furnace, and reaches all the way up to the crude ore. Soon after the crude ore is put in the partly desulphurized pyrites begins to melt, and runs down, as inside of little Bessemer vessels, acid lined, formed by each interstice. The melting sulphide trickles slowly down, borne on the blast; and in the high heat of the focus, in the joint presence of the air and the silica, are given the conditions necessary for the oxidation of the sulphide to the protoxide and its simultaneous union with the silica. The result is the slag; that portion of the sulphide which does not form slag forms the matte. The copper itself simply passes down from top to bottom of the furnace unaffected, and joins the matte. It is fully protected against oxidation and scorification by its great affinity for sulphur.

It is fair to assume that the volume of the masses descending is decreased very materially while in the focus, and that this is a region of great contraction of volume of charge, due to the upward escape of the gas formed and the consolidation of the solid pieces into slag. Even the volume of that portion of the sulphide which is not oxidized, but forms the matte, actually shrinks, for it continues to lose sulphur by sublimation.

Below the focus there is but little chemical action, and the furnace is full of inert silica. The focus action goes on above this supporting mass of silica, and below the focus the fused products occupy comparatively little space as they descend to the crucible. So they probably only occupy a narrow channel along the center line of the furnace, with an inactive thick lot of stuff on the walls of the hearth, which only slowly wastes away. When we blow out a furnace we see what appears to be a remnant of this stuff, the substructure of the focus, so to say, left behind in the hearth region. It consists of a honey-combed mass of quartz and slag, and is no trouble to remove. There is no matte visible, as it has been tapped out. Above these humps on the walls there may be the crude material of the last charges fed.

MR. EILERS.—Then the furnace in action never has bright tuyères?

MR. STICHT.—No. Bright tuyères are avoided. A light in a tuyère means that some matte is pocketed in that vicinity up against a jacket, and when this tuyère is punched the matte is likely to run out into the tuyère box, even though the tuyère is inclined inward, and cause a mess. Our tuyères are not circular, but long and oval, like a wide mouth, thus approaching a continuous tuyère opening all around the furnace as nearly as possible. But we still retain the old tuyère connections, such as a bustle pipe and downcomers from it to the tuyère boxes bolted on to the jackets over the tuyère openings. We have no valves on the downcomers or boxes, and no elaborate peep holes, simply a wooden plug in a round hole. There is, of course, a slag escape in the bottom of the tuyère box. All this is strong in cast iron, and I like to make the downcomers as short as possible. When we replace jackets we only do it in number, and then from the inside of the furnace when it is down. The tuyères are methodically punched several times a day, and should show no fire. It will interest you to hear that if you catch some of the gas coming out of the tuyères or through a tube inserted into the interior of the furnace through the tuyères, it is only common air, with no SO_2 in it.

MR. EILERS.—How long does it take to form the siliceous skeleton?

MR. STICHT.—To form it properly would take several days, and it might not then be just right, so we put it in purposely. We make it artificially.

ARTHUR S. DWIGHT.—Will you kindly describe the method of blowing in one of your blast furnaces? I have understood, in a general way, that with the blowing-in charge of coke you add only silica at the start and then gradually substitute the more fusible ingredients. As the usual blast-furnace practice is to make a blowing-in charge as fusible as possible, using a large proportion of slag, easily fusible ores, etc., your procedure is peculiarly interesting.

MR. STICHT.—We do. After the usual beds of fuel for heating and ignition purposes we put in loads of quartz up to about where the focus

begins. I dare say it is a layer 4 or 5 ft. high, and on top of this we start regular charges right away.

MR. ROGERS.—What percentage of fines under $\frac{1}{2}$ in. do you have?

MR. STICHT.—Sometimes we have 100 per cent. fines for a little while. I cannot tell you exactly. The fines question has not really troubled us at any time. The larger pieces compensate for the presence of a small amount of fines. Of course we must be careful not to have too much.

MR. ROGERS.—What about flue dust?

MR. STICHT.—That is the bane of our lives. We have done everything we could think of with it, briquetting, bricking, reverberatory fusion, etc., except to use the Dwight and Lloyd sintering method, which is still new, and it has all been too dear. We now simply wet it with water and throw it back again. That is a practice I do not wish the stenographer to record, however.

MR. WIERUM.—There is one point, and that is the efficiency of the oxygen you use. I would like to know whether there is any free oxygen in your escaping gases, and if not, why not, and how sure you are, and what would be the effect if you had any.

MR. STICHT.—There are the original 40 analyses made some years ago, which interested every one and which disclosed an oxygen efficiency of practically 100 per cent. They are still true. We have repeated them at intervals ever since, by different chemists, some of them Americans well used to the work, and the position is still the same. As long as we do proper work there is practically no oxygen going out; at any rate, nothing like 2 or 3 per cent.

MR. WIERUM.—Was that so with the old and lower furnaces?

MR. STICHT.—Yes. I built these as high as lead furnaces commonly were at that time, and higher than most copper furnaces. Copper smelters have been slow as regards furnace height and when they have wanted to do pyritic work they imitated the low column of the Bessemer converter. But this is a mistake, I think. A high column has given us a better Bessemerizing action than a low one, but the gases seem the same.

Like most people there, 20 or 25 years ago, we let our gas compositions take care of themselves, at first. It was only when we went into the question of making acid out of the gases that we analyzed them, and then re-discovered (what Hollway knew many years before) that the oxygen absorption was complete. This year I have had the tests repeated, and it was done with a water-jacketed apparatus which removed all possibility of the oxygen being taken up by the iron pipe, and which allowed us to go down to $7\frac{1}{2}$ ft. below the top of the column. We were always sure there was no danger of the oxygen going to the iron of the pipe for the reason that we have a tremendous evolution of metallic sulphur in all parts of the furnace, from focus up. If we allow a long tube of 2 to 4 in. diameter to descend with the charge we get a continuous stream of sublimated sulphur

passing off. It is evidently in a highly heated state, for it issues from the top of the pipe, even at a distance of 12 ft. above top of charge, in a transparent condition, and looks like a stream of hot oil. After shooting into the atmosphere about 2 ft. it condenses into a flame of burning sulphur with flecks of yellow sulphur vapor involved in it, which also take fire. Evidently the oxygen of the outer atmosphere is required to burn this excessive sulphur, and as far as the oxygen of the blast is concerned, it has had its fill of sulphur in the furnace column. The oxygen escaping unused from the focus has plenty of opportunity for meeting sulphur in the preparatory region above, where three-sevenths of the total sulphur on charge is liberated by distillation, being the so-called loose atom out of the pyrites. The latter here turns into an equivalent of pyrrhotite, but even this keeps on losing sulphur by sublimation in addition to oxidation.

MR. WIERUM.—Have you ever struck a balance of the total oxygen blown in with the total amount of iron oxidized?

MR. STICHT.—Oh, yes. We oxidize 95 to 96 per cent. of all the iron. As far as the sulphur goes, not so much is oxidized. The portion which is oxidized is only that which is retained in the final or ultimate sulphide which carries on the work in the focus. All the rest of the sulphur escapes as such, and the first oxygen it meets for combustion is that of the outer atmosphere above the top of column. Of course I am speaking only very generally, and making no allowance for particular cases. For instance, the presence of an oxidized ore will contribute to the combustion of some of this sulphur within the column. Then again, I am only speaking of our own work; where only pyrrhotite ores are treated the oxygen absorption may not be so complete.

GEORGE A. GUESS, Toronto, Ont., Canada.—The subject of pyritic smelting has interested me for some time. I have been engaged in the work for about eight years.

The information we have had in America regarding Mount Lyell practice has been rather meager. We knew a few of the basic facts. We knew of the 100 per cent. oxygen efficiency, and that a very low coke ratio was used. In American practice there did not appear to be anything like the practice in vogue at Mount Lyell. All the American plants doing pyritic smelting have had to use more coke, and it has been very interesting to learn to-night that the higher coke now in use at Mount Lyell has been a result of treating less pure ores, due to the exhaustion of the deposits of heavy iron pyrites, and that present practice at Mount Lyell is now more like the work in America.

The amount of oxygen in pyritic furnace gases was a subject of my attention while with the Tennessee Copper Co. There you know there is an acid plant. The Tennessee furnace gases contain oxygen. Some recent pyritic work I have been doing gave opportunity to again check the oxygen efficiency. Although we did not sample the furnace gases in this

case, the blowers were new, we knew what volumes of air they delivered, and, correcting for temperature and barometric pressure, an accurate calculation was possible. We were smelting an ore containing practically no sulphides but iron pyrites containing copper. There was little or no pyrrhotite. About 87.5 per cent. oxygen efficiency, I figured, was the result in this test; which extended over a period of 6 days when everything was running normally. Our furnaces at Tennessee, as you may know, were at one time dampered back so much in order to get a pressure on the gases going to the acid plant, that no false air could get to the furnace gases. This gave opportunity to sample the gases, which we sampled for SO_2 every 15 min., and at stated periods we examined for oxygen. We noted that when a charge containing a high amount of sulphur as pyrites was dumped into the furnace the SO_2 immediately went up. Now I cannot see how the SO_2 content could go up unless there were free oxygen in the upper shaft of the furnace. If no SO_2 were formed until the focus of the furnace was reached we would never have that rise immediately after the charge was dumped in. So in my experience I have never found a 100 per cent. oxygen efficiency.

It is interesting to hear this evening that Mr. Sticht has had these gas analyses repeated recently, and that the results have practically checked up the original finding. It is probable that his high oxygen efficiency is due to the high ore column which he carries.

Anyox is the latest addition to pyritic plants in America, and is already no infant. It will be one of the big smelting plants in America.

The furnaces are 30 ft. long. They were 50 in. wide at the tuyères, and had a uniform bosh from the sole plate to the feed floor. That was changed, the upper tier of jackets being set vertically 6 ft. apart. The lower jackets were moved out, so that the furnaces are now 52 in.

There seem to be no contractions in the pyritic charge until it reaches the focus of the furnace; and I see no reason why there should be a bosh in the jacket until somewhere near the focal point.

The System of Charging.—I believe that Mr. Sticht has a much better system of charging a furnace than is practiced in America if we except the little plant at Ducktown where the Freeland charging machine is used, which gives an ideal charge. The charge cannot go in in the careless manner used in the ordinary copper furnaces. If that careless manner is used it will form crusts and more crusts. I think we will have to pay more attention in this country to the charging of pyritic furnaces, particularly in order to get long campaigns and better running.

The Anyox Ores.—The mixture of the two ores contains nearly enough silica to flux the ores. When that mixture is being smelted there is not much latitude in the charge. If one has a clean pyrite, with a low silica, and uses clean siliceous material, clean quartz, there is better control over operations. One has cleaner material and less endothermic reaction in

his furnaces, with a hotter focus, resulting in a better slag. When the silica is combined with alumina, and the alumina in the ore runs up to 7 per cent. or better, and there is a good deal of schist, and the charge does not allow very much clean quartz, the range of operations is restricted, and the furnaces have difficulty in smelting and are likely to get into trouble.

The Anyox ores may give difficulty in their high alumina content; but so far 8 per cent. alumina in the slag does not give trouble. We ran several days with 8 per cent., or higher, but we have never gone to 9 per cent. I do not know how that would be.

Coke consumption is about 5 per cent. on the ore.

I would like to ask this question, Mr. Sticht: You were using 20,000 cu. ft. per minute in a 17½-ft. furnace; did you run with constant speed or constant pressure?

MR. STICHT.—The pressure varies. We run with practically constant speed.

MR. GUESS.—I think the constant speed is better.

MR. STICHT.—You can force the 20,000 cu. ft. up one blow hole, and in that case your pressure would go down. Now, it is more to the point to count the revolutions of the blowing engine, in other words, its delivery.

MR. GUESS.—If your pressure went from 55 to 65 or 70 oz., what would that indicate in your furnace?

MR. STICHT.—That it is too tight.

MR. GUESS.—How do you correct that?

MR. STICHT.—We would consider something ominous was about to take place if we kept on, but it would only amount to an eventual automatic stoppage of the furnace after slowing up, accompanied by an excessively high matte, up to 60 or even 70 per cent. in copper. But we would not try to loosen up the furnace with slag charges, as this would only be a way for wasting coke. We would run to the bin with the coarse ore.

MR. GUESS.—You think it is due to fines?

MR. STICHT.—Yes.

MR. GUESS.—Would you have 25 per cent. of fines in your total charge?

MR. STICHT.—Oh, no; that is a great deal.

MR. GUESS.—What would it be?

MR. STICHT.—Say perhaps 10 per cent. This difficulty has never troubled us. If one portion of the bin gets very fine, we go to another bin with coarser ore and equalize it. It is only the pyrites that sometimes gets fine, and some of the schisty North Lyell ore, and if the furnaces want a little nursing we nurse them. Fine schisty ore gives more trouble than fine pyrites.

MR. GUESS.—Most people that run blast furnaces like to have their

charge as free from fines as possible, and I thought they were particularly disadvantageous in the pyritic furnace at Anyox. This fall the crushing plant broke down, and the ore was bulldozed. We had a "dynamite rock breaker." Steel rails were set with openings of 13 in., and the rock was dumped on those and broken through with powder. The men did not break the rock smaller than necessary to get it through, and we had pieces go to the furnaces as large as a large suit case.

The ore was unscreened, and we had 20 to 30 per cent. fines. It was a case also where there was 5 per cent. moisture in the ore. It was a rather undesirable mixture to smelt. The stuff smelts all right.

We were running with constant air pressure, 1,000 ft. of air per foot of furnace length per minute. The pressure would be from 36 to 44 or 45 oz. Our slag was 37 per cent. silica; 42 or 43 per cent. FeO; 6 per cent. of lime, and $7\frac{1}{2}$ per cent. of alumina.

MR. LLOYD.—Below what point did you call your ore fines?

MR. GUESS.—One-half inch.

H. A. PROSSER.—Does the furnace run better after or before you put in these large lumps?

MR. GUESS.—I do not think large lumps of ore are an objection. As a matter of fact, we were getting the furnaces straightened around to run, and the crusher broke down while that was in progress, so I do not know that these large pieces are not helpful to the furnaces.

The method of charging can be improved upon. It was not possible to place the charge as well as it can be placed, and a crust was formed along the jackets, and as the crust built out the large pieces sometimes could not pass and formed a bridge. When that happened there was trouble.

T. T. READ.—Mr. Sticht has spoken of the question of the bosh. When you get a charge column 18 to 24 or 25 ft. high, as it descends there would be some shrinkage, but also some expansion due to heating, and then this might accentuate the trouble caused by fines. How about using boshes which slope outward?

MR. STICHT.—We might base our suppositions on the fact that there is no swelling. The pyrite does not swell; it first decrepitates and then melts 3 or 4 ft. below the top of the column, and then trickles down. In addition, it loses roughly one-half of its sulphur, and all this means a great reduction in its volume. The silica also decrepitates, and does not essentially expand with heat. The slag which forms out of the FeO and the silica certainly occupies less room than the solids did out of which it formed, so there is a contraction of volume all round, except in the case of the gases. The SO₂ does not occupy more space than the oxygen which created it, but both this gas and the other two, nitrogen and carbonic acid gas, being greatly superheated, doubtless exhibit a great expansive force bordering on the explosive.

Sometimes I have thought it might be well to build the furnaces this

way (indicating a wine-glass contour). There is a mass of *quasi* inert and stationary material in the furnace under the focus, below which the chemical action does not sensibly go, and which mass does no harm, and is not in the way. It carries all the charge. But no doubt the focus fluctuates up and down in its position, in accordance with the pneumatic variations inside the furnace, so you cannot fix its location to any permanent spot. By drawing in the profile suddenly to such narrowness the vertical movement of the focus becomes a source of danger, for it might easily happen that at times the focus action would be taking place right on the flat bosh of the jackets and lead to disastrous steam generation. In any case it would be a difficult construction, subject to troubles that readily suggest themselves. Furthermore, it does not seem at all necessary to depart from the present shape of furnaces, with nearly vertical walls, as long as the space inclosed by these walls is wide enough for the intended capacity, etc. In the space between there is plenty of room, at no constructional cost, for the furnace activity to comport itself as it pleases, and the presence of the inert humps is no disadvantage. As the actual furnace profile I like to regard not the casing of water jackets, but the space of shifting outline within the crusts formed on the water jackets, in which space the heating, fusion, and exodus of the products take place.

The question of bosh is of interest to the academic metallurgist, and sometimes bothers the practical metallurgist. But in our case the descent of the masses is so radically different from what it is in ordinary smelting that the conventional notions do not apply. We know the furnace forms its own bosh just as it forms its own slag. Using more coke, you can do anything; that is, you can approximate or fully establish the conditions of descent of ordinary smelting; but in our opinion the furnace had better be left to form its own bosh, and the coke omitted. I have heard of furnaces being built triangularly, with the base in the hearth, the bosh sloping outward instead of inward, but have no precise knowledge as to reasons and results.

MR. BROWNE.—In the shaft furnaces they have at Great Falls they are said to have had one with a belly on it.

R. H. VAIL.—Will Mr. Sticht tell us how the charges are made up at Mount Lyell and fed to the furnaces?

MR. STICHT.—I am afraid you will term us old-fashioned. We are not large enough to have some of the mechanical transportation and charging devices which have been brought out. We still take the materials to the furnaces by hand in charging barrows, or rather carts. We bring them along singly, and empty them out of the end of the carts, in a long string, along side of and close up to the edge of the furnace opening on to the charging plate. The coke is brought first and pushed in. Then all of the iron pyrites is shoved in. Then the siliceous ore is put in on top of that. Then the limestone, and then the slag, and that is the end of

it. Any flue dust, accretions, and things of that kind, are similarly put in. The feeding is done into both sides of the furnace. It is absolutely necessary for us to have the greatest elasticity in the delivery of the materials to the furnace. All our success in permanently working to a high concentration ratio and avoiding double smelting depends on this and proper feeding. Conveying appliances from bins to furnaces cannot do what our furnace crews have to do, *i.e.*, use judgment in the selection and placing of the materials at the furnace. We have exhaustively gone into the question, and cannot find that we would either cheapen or improve our work by substituting mechanical delivery and charging for our trained wheelers and feeders. We tried a self-dumping charge car, of a clever construction, for a long time, but finally proved it a mistake. The only mechanical device we use is a feeding machine. The feeding used to be done by hand, by simply pushing the materials flat over the inner edge of the plate. But with the increased blast pressure this work got too hot and we had to increase the number of feeders. So we invented an "iron man," which consists of a framework embracing each long side of the furnace, and which slides and rolls on the charging plates in such a manner that a line of hinged pusher plates in front, and parallel to the whole length of the charge opening, gently but strongly shoves the materials over the edge, the same as the feeder's shovel used to do. The framework feeds the furnace alternately on each side of the furnace with full charges, and it is an easy matter to allocate the materials on top of charge in any desired manner. The feeder simply directs the wheelers where to place them on the plate, so that he can feed the furnace with such discrimination as blow holes and unequal descent of the top may demand.

Our metallurgical crews have developed a fine sense of feeling as to just how and where the stuff should be put in. The inner edge of the charging plates projects into the furnace with a certain overhang. Much discussion was had as to whether this overhang should be 9, 10, or 15 in., so we invented a movable charging plate to meet the case. Generally speaking, it does seem necessary to let the charge fall 10 in. from the wall. The relative distribution of coarse and fine on top of column depends on this measurement and the height of drop. The latter varies, as the furnaces are not always held full to the top.

Our general practice, modified by many side considerations derived from experience with particular kinds and varieties of ore, is to feed so that the center line of the furnace is porous, and the blast does not prominently come up on the jackets. We like to see the blast rise out of the column evenly all over, and this is not an easy matter with pyritic smelting. I am sure our masses do not descend in the dish-shaped layer style figured in the handbooks, but there is a general jumble or mixture without stratification. As there are no layers of coke fuel the ignition cannot be regarded as taking place in successive stages as the charges come down to

the respective horizons—though this is too cramped a view even for ordinary smelting. The first iron begins to burn on the upper periphery of the focus, and combustion and slag formation are intensest in its lower portion, without doubt.

MR. LLOYD.—Is it necessary to keep the jackets cold or warm?

MR. STICHT.—They should be kept the same as with any blast furnace, but as a matter of fact we run them very cold.

MR. LLOYD.—Does all the converter slag go into the furnace?

MR. STICHT.—Yes, it is thrown together with whatever blast-furnace slag there may be from settling pots, etc. The blast-furnace slag is all granulated, which was done from the first, and, although it is a well-known and simple operation, it was considered one of the great triumphs of the human mind over matter. Water is too short in other parts of Australia for the idea to have arisen there. To the local layman it seemed miraculous that water could wash red-hot slag away without causing a deadly explosion. In the case of the matte, it did not take the men long to find out the difference.

The smelter men we found very eager to learn everything about their work, for they felt that, instead of having to remain farm hands, they were learning a new profession, which brought them in better wages and a nicer life. The furnace hands are exceedingly clever and resourceful, and it has been a great pleasure to work with them. Of course the opportunities for finding employment are not numerous, so a man behaves well to hold his job down. If we dismiss a man he is very likely to be back on our hands again in six months, having made the rounds of the island without success.

On the whole, all the men are extremely satisfactory, except perhaps the miners. The miners show a fair efficiency, but lack the snap of the American miners. The work in our mines is done on a contract basis as far as possible, not task work, but regular contract, the men bidding in parties. We pay by the smelter dry weight for the ore they stope. The ore gathers its moisture on the way to the smelters and the miners do not supply that. The average annual rainfall is 110 in., so it must be taken into account.

MR. PROSSER.—My experience in pyritic smelting is very limited. The reason I asked Mr. Guess about the action of the furnaces after the crusher broke down, is this:

A certain plant was preparing to start operations on a pyritic basis with an able metallurgist whose experience had been chiefly in lead smelting. The crushing facilities were limited, so some lumps of ore were 10 to 12 in. in diameter. He was very much afraid that these lumps would cause trouble. After starting the plant, it was the general opinion that these lumps were not at all detrimental. Perhaps the breaking down of the crusher mentioned by Mr. Guess may have been a blessing in disguise. In the case mentioned the siliceous ore was pure quartz;

the sulphide ore ran 4 per cent. of copper. Under ordinary conditions we could make a 30 per cent. matte, with 2 per cent. of coke in the charge. If for part of the clean quartz an aluminous ore were substituted (containing say 70 per cent. SiO_2 , and 16 per cent. Al_2O_3) then we would have to put double the percentage of coke on the charge, and the matte would run only 15 per cent. copper. The two most important causes of the trouble were alumina and zinc, either of which would lower the grade of the matte and increase the amount of coke required. The other necessity was to avoid an undue proportion of fines.

MR. LLOYD.—I asked a question with regard to the fine ore. The expression has been used many times and I asked Mr. Guess what he called fine ore, and got the answer from him "below $\frac{1}{2}$ in." This question has been discussed by many people; it has struck me that the name "coarse ore" or "fine ore" depends on how the ore acts under heat. I have had cases where $\frac{1}{2}$ -in. ore would be beautifully coarse ore; and again I have had cases where larger than $\frac{1}{2}$ -in. ore would be called "fine ore."

What I want to say is that the sulphides act differently in different cases according to how they are constructed or crystallized—the roughly granular stuff will break down quicker than finely crystallized stuff. And if the ore did not break down under heat, and was not fine, one would not use such large chunks. In other places by decrepitation coarse ore might become fines and the fines still finer.

MR. GUESS.—Speaking in reply to that, Mr. Lloyd, I remember I wanted more iron one time, and I got a shipment of heavy iron sulphides—iron pyrites that contained low silica. It was all 3-in. stuff. When it was put into the furnace there were explosions that were like musketry shooting all over the place. The result of the rapid decrepitation was that the ore went into pyrites sand and ran the furnace pressure up. Decrepitating ores have no place in a blast furnace.

J. B. F. HERRESHOFF.—I saw the work at Anyox, before Mr. Guess went up. I asked Mr. Guess what he did, when he came back from his trip. He said, "I cannot say that I did very much, the ore was suited to pyritic smelting;" but I think he did make some changes up there which were vast and efficient.

But Mr. Guess will not object if I say that before he went up they were pouring the converter slag into the receptacle. Mr. Guess had some of the slag chilled, and put back into the furnaces. He paid particular attention to the charging of the furnace, and found the fine material in the ore bothered by improper distribution in the furnaces.

When I was at Anyox the difficulty was with the furnaces. The blowers gave 15,000 cu. ft. of air to the furnaces, and the pressure was about 48 oz.; when the furnaces ran at high tonnages, the pressures dropped to 36 in., and they got 30,000 cu. ft. through.

MR. PROSSER.—In relation to the charging of the coke, Mr. Sticht, is that done through the side doors by the iron man?

MR. STICHT.— Yes.

MR. PROSSER.—In Mr. Jennings's plan it was considered advisable to charge by hand. In fact, it was charged through the end doors by hand. Both furnaces have end doors.

MR. STICHT.—We have end doors for barring purposes only. If we wish to put all the coke in one place with the mechanical feeder we can do so. If the furnace attendant wants to put it elsewhere, he has it dumped so for the "iron man."

It may interest you to know how our furnaces are run—namely, backward. We do not figure out charges, as it is of no special use for immediate running purposes, but manipulate according to the grade of matte which falls. The quantities of pyrites and of limestone are kept constant on the charge. We vary only the siliceous ore and the coke, but mainly the former. Now if the matte, which we want to have 45 per cent., only goes 30 per cent., we put more siliceous ore on, and we know beforehand that a certain charge composition will bring it up to 45 per cent. in not more than $1\frac{1}{2}$ or 2 hr. For this control frequent matte assays are necessary, and in order to disburden the assay office we have taught the shift bosses to take a dip sample four or five times in a shift and analyze it for copper. They may not know anything of chemistry but they are attentive and careful. From time to time they receive a bottle of standardized cyanide of potassium solution from the laboratory with the *titre* marked thereon by the chemists, and they weigh out a certain amount of ground up matte on a pulp balance and dump that into a beaker glass, put a little nitric acid on it, boil and dilute with water, add ammonia, stir, and titrate direct from an ordinary burette until the color disappears, filter, and titrate again to a finish. They then read on the burette the numbers of cubic centimeters used, and multiply by the figure given on the cyanide bottle, and, if they do not happen to know decimals, they take the first three figures from the left, the third being tenths. Their determinations are near enough, within 1 per cent., to guide us in the matter. It is a simple thing and easy for these men to do. It makes the furnace work more interesting for them.

This idea of determining the copper in the matte currently by the shift bosses was introduced by Mr. Beardsley. He gave them a "bush" balance constructed of bits of wood, glass rod, wire, a needle and paper, which worked quite as well as the present pulp balance (when the wind was not blowing). Of course these tests are only for running the furnaces. We follow a thorough system of mechanical sampling of all ores and products, and keep elaborate records.

MR. BROWNE.—To listen to the speeches made here, one would think that pyritic smelting was a sure cure for all the ills that smelting is heir to.

I wish to warn you against too great optimism. Now this method is not by any means a panacea for all our evils. It is not like the celebrated nostrum guaranteed to clean the teeth, sharpen skates, and cure that pain in the back upon rising.

There are certain ores upon which this much-vaunted process will not work. I think Mr. Sticht will bear me out when I say that it is easy to get a process but often very hard to get an ore suited to such a process. In our own case we have an ore which contains about 35 per cent. iron, and 25 per cent. sulphur, with 18 to 22 per cent. silica. Theoretically such an ore should smelt pyritically. If you put this ore in a furnace with about 8 per cent. coke and enough lime to flux the silica, you can get this charge to smelt easily and make a low-grade matte. Now, following the Tennessee practice, the next step would be to cut the lime off gradually, charge by charge, and replace it by silica in increasing amount, until the furnace was operating on a mixture of quartz and ore. I have tried this with every variation I could think of and have not yet been able to effect a true pyritic reaction. The ore undoubtedly contains enough heat units latent in its iron and sulphur to make it work pyritically, but for some reason it will not so far work.

The matte formed in the presence of silica is low grade. It seems to liquefy from the ore, leaving behind what Mr. McArthur used to call "the gummy mass before the tuyères." This mass in time blocks the furnace and drives the heat to the top.

In order to get the heat out of this matte and flux this mass it is necessary to burn the iron contained in the matte. But once the matte has trickled down past the tuyères you can get no more oxidation, and no more heat.

In the Knudsen furnace the matte is held above the tuyères and there forced to oxidize and to combine with the siliceous skeleton or gummy mass out of which it has melted. Hence in the Knudsen furnace the problem is determined by the mechanical position of the matte.

If in the blast furnace the low-grade matte could be held back above the tuyères, it would undoubtedly give out its heat and form an easy-running slag with the rest of the ore.

This problem is one of our future plans. In the past many have tried it without success. Some day it will be effected, and I will take off my hat to the man who tells me how to do it.

Progress in Metallurgy

BY JAMES DOUGLAS, NEW YORK, N. Y.

An address before the Meeting of the New York Section, Nov. 4, 1914.

As life advances one is inclined to look backward instead of forward, and the vista over which my memory carries me has been filled with such a shifting panorama of changes in metallurgy that the whole practice of the art seems to have been re-created.

The first cupola furnace that I saw in operation was at one of the group of mines at Capelton, Quebec, from which the Nichols company still draws some of its sulphur supply. At that time it was operated by a Hartford, Conn., company, under a General Adams. When its small brick furnaces made a campaign of a week and smelted 10 tons a day, the feat redounded to the credit of the operator.

The introduction of the water-jacketed cupola dates forward to the next decade. I was using one in Pennsylvania, making copper matte, to the horror of a noted metallurgist, who could not conceive it possible that you could bring fused sulphides in contact with steel without rapid corrosion. In our ignorance we were doing what seemed to be an impossibility, and it is just that recklessness and disregard for precedent which has characterized so much of the work of American metallurgists, and carried us forward at such headlong speed.

Once the water jacket was accepted as the type of the copper cupola, its almost unlimited expansion in length, but not in width, was a matter of convenience rather than of skill. In Butte the cupolas have been enlarged by adding furnace to furnace.

The first reverberatories which I saw in operation were those in Professor Hill's Black Hawk establishment, where the concentrates from the Gilpin County gold mills were run down into matte for separation in England. Sixty dollars per ton was the usual deduction in those days before gold and silver was paid for. Smelting was done with wood as fuel, and the capacity of the furnace was about 10 tons per diem. Shortly after this Professor Hill formed an alliance with Richard Pearce, who was running what, if I recollect aright, was called the Swansea Works, at the bend of the river approaching Georgetown.

Mr. Pearce had been engaged in the separation of precious metals from copper products in Wales, and applied his skill to the local separation of the Black Hawk mattes. Soon the operations were shifted to the larger works at Argo, near Denver, where for many years the Ziervogel method was practiced with much success. And to Pearce is due the credit of taking the lead in gradually expanding the size and capacity

of the reverberatory, for at Argo and at the company's branch works at Butte this type of furnace grew rapidly in dimensions to 35 by 16 ft., and a capacity of 50 tons.

Big as these were, they were pigmies compared with the huge reverberatories of the present day, heated by oil or coal, smelting as much as 660 tons of charge a day, and fulfilling what the old metallurgist dreamed of as a possibility, but failed always of securing as a reality—the recovery of the waste heat in the form of power. (Mathewson's *Development of the Reverberatory Furnace* and my Cantor Lectures.)

The fate, however, of the Argo works, despite the science and experience which Pearce brought to bear upon his operations, was virtually sealed when the combined forces of the pneumatic method and the electrolytic separation came almost simultaneously to the aid of the metallurgist. The converter brings the matte to metal in a space of time measured by minutes. Electricity at one operation refines the copper and separates the precious metals, simplifying the process and saving time and money. The Argo works had therefore to go out of blast, and the metallurgist retire before the engineer and the electrician.

And so one has watched change follow change, machinery in every case taking the place of hand labor; and, inasmuch as mechanical force can be generated to an almost unlimited degree, the rate of production has kept pace with the contrivances for generating and applying power. Thus it has come about that the trifling amounts which were made within the memory of man have increased to the stupendous production of to-day.

The changes have not been confined to smelting, but have been many and very conspicuous in the concentrating department. Isolated motors have in great measure abolished the countershafts; and grinding mills of many different makes have competed for acceptance in rapid succession. The National riffle has even displaced the jig in some mills, and now flotation is claiming to supplant all other methods. Whatever invention may for the time being claim supremacy, the experience of the past compels one to admit that the construction of the mill of the future should be so elastic that it can be altered with the least expense and embarrassment to suit the many changes which will become imperative. It will probably consist of a large square structure, within which the devices temporarily employed will be erected upon platforms and scaffolding which can be readily removed and replaced.

We have been recalling incidents in the history of copper metallurgy, but it is the growth of the iron industry which excites our imagination and wonderment. Its consumption is really the barometer by which we can gauge the world's activity and the demand for other articles; for there is an economical relation in the use of one metal and another. As the demand for iron is the controlling factor, not only in the metallur-

gical world, but in the industrial world at large, it is iron which fixes the rate of production of virtually all the other metals. I pointed out some years ago that the relation of the consumption of copper to iron in this country was as 1 to 83. That relation has been maintained ever since, and holds good approximately in the world's consumption of iron and copper. That being the case, the demand for the metals, and therefore to a certain degree the price which they will command in the market, depends upon the economic laws, which will inevitably override our attempts at interference. When these laws come to be better understood and submitted to as implicitly as we bow to the laws governing the forces of nature, the large groups of people who compose the industrial world, instead of engaging in rivalry, which is liable to degenerate into hostility, may possibly co-operate to advance the interests common to all. A world combine would be clumsy to handle, but a world's congress might pass international laws for the regulation of trade which would obviate some of the anomalies that exist to-day.

H. W. HARDINGE, New York, N. Y.—I wish to acknowledge an unsettled credit due to Dr. Douglas of which he is probably not aware. When Dr. Douglas visited the Arkansas Valley Smelting Co.'s Leadville plant about 25 years ago, I was manager. A casual remark at the time by Dr. Douglas was the basis of certain changes in smelting operations through the conversion of a lead stack into a composite lead and copper furnace.

The furnace slags at that time, owing to a shortage of lead as a desilverizer, were running in the neighborhood of 3 to 4 oz. of silver per ton. Taking advantage of the suggestion by Dr. Douglas, as well as certain experiments already made by Herman Keller, the Superintendent, $\frac{1}{2}$ per cent. of copper, in the form of ore, was added to the charge. The resulting slags immediately dropped to less than 1 oz. of silver per ton. Later that portion of the slag dump, amounting to several thousand tons, which had been assaying unusually high, was taken into stock as an asset on the order of the President, A. R. Meyer.

At the time of these excessive losses in silver (which was when silver was selling for about \$1 per ounce) the profit and loss balance had for several months been in "red," but within two months from the time the addition of copper was made to the charge, the monthly balance turned to a profit of \$5,000, and to the best of my recollection a five months' average slag content of silver was 0.93 oz., and the balance for the fifth month changed to \$17,000 profit; thus a casual remark resulted in the changing of copper smelting in Colorado, for up to that time the by-products of the lead furnaces in the form of copper mattes had been shipped from the State to the Orford Copper Co. and other refiners in the East and abroad. Shortly after this other smelters in Colorado adopted the same or similar methods, and within a year not a pound of

the lead-stack copper mattes was shipped out of the State. The re-smelting of these slags, which had a good iron content, admitted of utilizing ores high in silica and at a correspondingly high smelting charge.

Previous to these and other experiments, lead-furnace mattes contained upward of 5 per cent. in lead, which was a detriment to the copper refining in the East, and for which a penalty was charged.

One of my colleagues in commenting upon the production of lead and copper in the same stack stated that it was impossible. This may have been a very well based opinion, but during the discussion there was a check upon my desk for \$10,000 in payment for a shipment of this impossible product of lead bullion produced from the copper stack. The high-grade copper matte made in conjunction with the lead had been shipped to the Argo works, near Denver, where the silver and gold contents were extracted in reverberatory furnaces in conjunction with Dr. Pearce's "secret" method, which was also a subject treated in Dr. Douglas's previous remarks.

Finishing Temperatures and Properties of Rails

Discussion of the paper of GEORGE K. BURGESS, J. J. CROWE, H. S. RAWDON, and R. G. WALTENBERG, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2433 to 2437.

ALBERT SAUVEUR, Cambridge, Mass.—The Bureau of Standards has once more rendered a public service. Manufacturers and consumers of steel rails alike should welcome the important results now laid before them by Dr. Burgess and his co-workers. Personally I welcome these results all the more that in the past, as some of the members may remember, I have never failed, when occasion arose, to call the attention of rail manufacturers to the importance of controlling the finishing temperature of their rails. The pioneer metallographic work conducted at the South Works of the Illinois Steel Co. in the early '90s clearly brought out the close relations existing between the manufacture, properties, and finishing temperature of steel rails and I think I may say without fear of being contradicted that the results of this early investigation had considerable influence in drawing the attention of manufacturers to the importance of finishing temperature and of heat treatment in general. The results now before us should be, it seems to me, of great value to rail manufacturers in their efforts to produce a better and, above all, a safer rail. It seems to be demonstrated in this paper that there is no difficulty in installing in rail mills pyrometric outfits, making it possible to record, without interfering in the mill operations, the temperature of the rails as they leave the finishing rolls, or of the ingots as they are withdrawn from the soaking pits, or at any intermediate point in the process of manufacture. The inference necessarily follows that manufacturers should see to it that such pyrometric installations are made, in order that they may actually know, instead of guessing at, the temperature of their rails; and, for obvious reasons, rail consumers should insist upon such installations.

In the rolling of heavy sections especially should care be taken to lower their finishing temperature, seeing that the present tendency is to finish them at higher temperature, whereas, owing to the slower cooling of their central portions, which makes for coarseness of grain, they should, if anything, be finished at a lower temperature than lighter rails.

It also seems as if the authors had demonstrated the utter uselessness of the shrinkage clause as a means of regulating finishing temperature. When we are shown that the shrinkage clause allows rails to be finished at a temperature as high as $1,120^{\circ}\text{C.}$, or some 450° above their critical range, it is high time to discard it, not only as useless but actually as dangerous.

R. TRIMBLE,* Pittsburgh, Pa.—One of the most perplexing things we have had to do with in connection with the manufacture of rails is to secure their rolling at a proper temperature, and, as stated in the paper, the railroad engineers have endeavored to have rails rolled at a lower temperature, thinking that such rails would give better service, while on the other hand the mills are favorable to a high temperature, on account of the reduction in wear and tear on the mills due to rolling the rails at a higher temperature.

Although we have in our specifications a shrinkage clause, which we think should be readily met by the mills, yet one mill insists that it is impossible for it to reach our shrinkage clause, and we therefore have to make an allowance to this particular mill, which is objectionable.

WILLIAM R. WEBSTER, Philadelphia, Pa.—The authors have given us some very valuable information on temperatures at which rails are rolled at our mills in every-day practice in the execution of orders under requirements of ordinary specifications. The results of their work will, as a matter of course, lead to a very thorough and complete investigation of the whole subject. The Bureau of Standards is well fitted to undertake this work and the members of this society should co-operate with them in every way possible.

The present paper forms a very good reopening of our former discussions on the physics of steel and it is hoped that the authors will favor us with further data, from time to time, in order that definite conclusions may be arrived at on the several points they have raised, and on others they have not referred to, but which are well recognized as having an important bearing on the subject and will undoubtedly be brought out in the discussion.

The practical mill men should be consulted and their views obtained in this investigation, as it is more complicated than at first appears. For instance, the suggested plan for controlling and securing the best finishing temperatures in rolling cannot be carried out in practice, owing to the section of our T rails. In rolling rails direct from the ingot, the ingot is necessarily hotter than is desirable, in order to carry the heat through to the final passes and avoid finishing the flanges too cold. In rolling rails from reheated blooms, it is impossible to secure the beneficial effects of such reheating, as the blooms are heated higher than is desirable, in order to carry the heat through to the final passes. This investigation will develop that it is not merely that the manufacturer "prefers to roll his rails hot, as less power is then required to operate the mill and there is less wear and tear of the rolls," but that with the present sections he is not only compelled to roll hot, but he cannot finish the heads at temperatures low enough to produce the best structure in the steel.

* Non-member.

It is therefore suggested that the authors repeat their observations on similar lots of rails rolled at all four mills referred to, under their own supervision, in order to see at just how low temperatures they can start with the ingot or bloom and how cold such rails can be rolled to maintain section and comply with drop-test requirements. At mills C and D, where four saws are used, the shrinkage of each rail should be measured in order to compare the variations with the observed temperatures of each rail taken by the pyrometer. It would also be well to note the amount of camber put in these rails in order to see what effect the cambering has upon the shrinkage and to determine definitely what the coefficient should be for both open-hearth and Bessemer steels.

The authors assume that their coefficient of contraction is correct and, based on it, state that the present shrinkage clause allows "rails to be finished at 1,120° C., or 450° C. above the critical range of rail steel." They also state, in connection with their own work: "From the thermal data, it follows that all these rails could have had work done on them to advantage during the rolling process down to 700° C. (1,292° F.), whereas for none of them was the actual finishing temperature lower than 880° C. (1,615° F.)." This finishing temperature of 880° C. at which some of the rails were finished corresponds to a shrinkage of about $5\frac{7}{16}$ in. in 33 ft., when figured by the same coefficient of contraction. Did the authors measure the contraction of any 100-lb. rail finished at this temperature, and if so, what was the actual shrinkage? The finishing temperature of 700° C. would only give a contraction of about $4\frac{3}{8}$ in. in 33 ft. when estimated by the same coefficient. Do the authors consider that any rails of the present sections can be rolled at low enough temperatures to give only about $4\frac{3}{8}$ in. shrinkage in 33 ft.? We have here the desired finishing temperature on the one hand and on the other the rolling difficulties owing to the present sections of rail. How are these to be brought into accord without adding metal to the web and flanges to permit the rails being finished at the proper temperatures? How much metal should be added to the web and flanges and how close would such modified sections approach to the English bull-headed rail?

The writer's views on this phase of the question were given in 1900 in an informal discussion at the annual convention of the American Society of Civil Engineers, in London, on Recent Practice in Rails. In March, 1901, he asked the Society to appoint a committee on rail sections to consider modifying the Society's sections by putting more metal in the flanges to carry the heat and allow lower rolling temperatures, and claimed that the rolling temperatures were of as much importance as the chemical composition. His views were also given in a supplemental report to the final report of that committee in 1910, and again in his recent Discussion on Rails before the American Railway Engineering Association in March, 1913.

Too much stress cannot be placed on the influence of the finishing temperature on the quality of the steel; but formerly its effects were not well known or appreciated. The writer's attention was first called to the great influence of the initial rolling temperatures, amount of reduction in each pass, and finishing temperatures, in his investigations on *The Relations between the Chemical Constitution and Physical Properties of Steel* in 1891 to 1894. It was then found that the control of the rolling temperatures was the most important factor. Since that time the writer has been greatly interested in the effect of the heat treatment on all grades of steel in forging and rolling and, as far as he knows, was the first to propose the shrinkage clause in rail specifications. He has always appreciated its limitations and the force of the objections raised from time to time, but does not consider that any practical substitute has yet been offered. It was the first effort made by the purchasers of rails to systematically control and check the finishing temperatures of their rails. The rail-mill practice prior to that time has often been referred to as "hardening the steel by the addition of carbon and squirting it through the rolls in order to get increased output." It is hoped that the authors will give us further data showing how accurate a measurer of temperature the shrinkage really is, and what reductions in the present specified shrinkages can consistently be made for rails of present section.

Assuming the pyrometer can be used as suggested, in every-day practice, no explanation has yet been offered as to how the individual rails on which observations have been taken can be identified in the finishing department; but the shrinkage of the rail leaves a permanent record. All the inspector has to do at mills where four saws are used is to measure the distance between saws and the lengths of the cold rails. This check on the finishing temperature by the shrinkage is therefore automatic in every respect.

There are two problems that come into this matter: First, public safety; second, wear of our rails.

If you roll your rails of present sections too cold, endeavoring thus to secure better wear, the flanges will be finished too cold, causing the steel to tear and thus forming small fractures, many of which are covered up in rolling and are not observable on inspection. These incipient flaws in the flanges of our high-carbon rails are much more dangerous than is generally appreciated. We should therefore go very slowly in this matter before coming to any final conclusions on cold rolling of rails until sections are decidedly changed.

P. H. DUDLEY, New York, N. Y.—

1. *Introduction*

The periodic recurrent discussion of one or more phases of rail manufacture emphasizes:

1. That the service of rails has increased in the track in a faster ratio than generally realized.

2. The complicated problems of rail manufacture for present service in the United States are but partly encompassed by metallurgic knowledge and practice, but include those of mechanical and civil engineering, operations, climatic conditions, and a combined mill and track experience.

Page 4¹ of the Introduction states: "It is common knowledge that there are an alarming number of failures of rails in service on the railroads of the country, and, unfortunately, these failures are not on the decrease. Thus the reports of the rail committee of the American Railway Association show 36,641 rail failures, in 12,688,714 tons laid, for the year ending October 31, 1911, and 61,047 failures for 13,736,956 tons [October 31] in 1912."

The increase of failures from 36,641 in 1911, to 61,047 for an addition of only 1,048,242 more tons reported (October 31) in 1912, revealed the fact that there was some decided change in the conditions of operation which contributed to lessen the duration strain factor of the metal in the rails to a greater degree for the low temperatures in 1912, than for a milder period in 1911.

The winter of 1911 was mild for most of the northern railways, but 1912 was extremely cold, with the greatest number of broken rails in their history; 1913 was mild, with 60 per cent. reduction over 1912 of broken rails, while 1914 was again cold in the eastern portions of the United States and Canada.

November, 1911, was cool, with a range of 50° to 120° deficiency temperatures for the month at different U. S. Weather Bureau stations. December had an opposite oscillation of warm waves of 100° to 200° above normal. Rails which had contracted in the splice bars during November, were again expanded in December. The sudden excessive cold wave in the West of the last one or two days of December, and the first day of January in the East, contracted the metal in the rails and set up tensile stresses before the rails could render in the splice bars. This, combined with the stresses of the metal of the rails as girders under the moving wheels of the locomotives and cars, was the important factor in 1912 of the greatest number of breakages of Bessemer rails in the history of the railways.

The ordinary duration strain factor of the metal of more Bessemer rails than usual was reduced from the effects of the cold by:

1. The additional tensile stresses due to being held by the splice bars from contraction.

2. The increased tendency of the metal of Bessemer rails of 0.10 phos-

¹ *Technologic Paper No. 38, U. S. Bureau of Standards.*

phorus and only 0.50 carbon to fracture under rapid distribution of the alternate unit fiber strains and balanced stresses of the moving wheel loads.

I sent out September, 1913, 450 copies of the tabulations of the accumulated excess or deficiency temperatures for the United States Weather Bureau stations on the New York Central Lines for the years 1911, 1912, and 1913 to Aug. 31, to the respective officials in charge of maintenance of way and to those of equipment.

I made the forecast September 1, that the probabilities were that the winter of 1914 would not be as mild as the winter of 1913—the broken rails not being one-half as many as in 1912 for the same locality and service.

I also sent instructions to have the maintenance of way to its highest standard in 60 days, and the equipment, as to the rotundity of the wheels, in the next 90 days, and to be maintained through the winter.

The forecast proved true, though December and the fore part of January were mild; the cold wave was severe in the latter part of January, February, and March, but from the care and attention of the officials of the respective departments, the percentages of broken rails and shelled wheels were less than in normal years.

The excess or deficiency temperatures per month for 1914, to August 31, have been added to the three preceding years, with forecast and instructions issued for the winter of 1915, and copies sent to the officials in charge of the departments of maintenance of way, of equipment, and operating. These three contribute by their care to the safety limits of the duration strain factor in the metal of the rails.

The large railway systems keep their own statistics up to date, therefore know from the comparative figures that the rail failures have decreased since the extreme cold winter of 1912, and are proportionally less than ever before for a similar service. I speak particularly for the New York Central Lines, for in some territory they often have cold waves of some days' duration of 30° and 40° F. below zero.

The introduction of basic open-hearth rails the past few years in which the phosphorus content is practically one-fourth of the former Bessemer rails, furnishes the metal of the sections as girders a duration strain factor of greater combined capacity and rapidity of distribution of the alternate unit fiber strains, balanced by equal stresses, than was possible to secure in many of the 0.10 phosphorus and only 0.50 carbon Bessemer rails.

The basic open-hearth rails for the New York Central Lines have all been made since 1910 under the 1909 specifications of their elongation and exhausted ductility tests which were developed from the elongation tests under the drop used for Bessemer rails from 1890 to 1910. There were in the tracks of the N.Y.C. & H.R.R.R. Jan. 1, 1914:

Tons	Basic Open-Hearth Rails		Length, Miles	Broken per 100 Miles
65,297	5½ in.	80 lb.	519	1.93
83,268	6 in.	100 lb.	530	0.57
67,460	6 in.	105 lb.	408	0.00

There were 13 broken basic open-hearth rails from Jan. 1 to Aug. 1, 1914, and 279 Bessemer, or 1 open-hearth to 22 of Bessemer, exclusive of interior transverse fissures. February and March were extremely cold, with many continuous cold waves of 35° F. below zero and an occasional wave of 40°.

2. Importance of Limiting Temperatures of Rolling Rails

This is a comprehensive subject and will require many papers and discussions to state its many salient features, for application to rail manufacture for the girder and resistance to wear.

I state here that the effects of the rolling temperatures upon the duration strain factor of the metal of the basic open-hearth rails as girders, for service either in warm or cold climates, is one of the keynotes of this discussion. Investigations have shown that the transformations of all the different metallographic forms the metal should pass through from the higher temperatures of the hot set ingot to the lower of the metal of the rail in the finishing pass, did not completely occur in the metal of the section, particularly of that in the central portions of the head. It is left in a heterogeneous condition in a few rails with initial strains, and as the interior metal cannot elongate as readily as its exterior envelope of the head, is sometimes checked by the gags of the presses to straighten the rails. These develop in service from the negative strains in the wheel spacing of the passing equipment.

3. Rolling Rails

Rails are rolled:

1. Direct from the equalized, original heat of the ingot to the finished rail.
2. The ingots are bloomed from their equalized, original heat, then the blooms are reheated to pass through the roughing and finishing roll trains.

The investigations show that as a rule the transformations seem more complete in a greater majority of rails from the reheated blooms, than in direct rolling. This is a difficult investigation to make direct upon the output of the rails owing to the small fraction of 1 per cent. affected. The fact is confirmed by only an occasional failed rail from the reheated blooms due to the incomplete transformations.

There have been two distinct schools of theory and practice for the manufacture of rails:

The first considers that the chemical composition should be prescribed and limited by definite proportions to make sound hot ingots, combined with rolling temperatures to secure the physical properties intended in desired sections of girders.

The second considers the chemical composition secondary, but that the rails must be rolled cold to insure good wear, as the only information to be given the manufacturer to secure a suitable product.

The first school has service tests of rails in the tracks for over two decades which proved safe and quite free from breakages in low temperatures, with fair resistance to wear. This has been of great service in the adaptation of basic open-hearth steel to replace Bessemer.

The second school had the mills remodeled for cold rolling of Bessemer rails (some were built for the purpose); and prescribed shrinkage limits for the hot saws, which have not been modified for basic open-hearth rails.

The holding the bars 40 sec. before the last pass, was abandoned many years ago, when the epidemics of broken rails in the low temperatures of 1904, 1905, and 1906 occurred. The mills which were built for direct cold rolling for wear, without the least intention of so doing, injured many rails of their output of basic open-hearth rails as girders.

The second school is no longer advocating the extreme cold rolling, for the service tests have not sustained its contention.

The metal of a rail section, as an element in the "Means of Transport" to carry and guide the passing equipment, performs two diverse functions:

1. That of a girder, to distribute in fractions of a second large alternate unit fiber strains, and which, for safety, governs the quality of metal which can be used.
2. That of resistance to wear and abrasion from the passing wheels of the equipment.

4. Interior Transverse Fissures Due to Incomplete Transformations of Metal from the Higher Temperatures of the Ingot Metal to That of the Finished Rail

It does not seem to matter what the distribution of metal is in the section, whether of thin or heavy bases, for each has developed interior transverse fissures in the head after more or less service. A direct connection seems to be established with cold direct rolling temperatures in retarding the transformations, which must receive careful research before it can be determined what is the proper range of finishing temperatures for the combined functions of a given section and its service. It may fail from an interior transverse fissure either in a warm or cold climate.

The paper is a valuable contribution of data, and I concur with the authors that more experimental work should be done, and the rails subjected to service tests to establish the important facts as to the finishing temperatures and properties of rails.

GEORGE B. WATERHOUSE, Buffalo, N. Y.—The paper of Dr. Burgess and his associates is one of great interest alike to the theorist, mainly concerned with pyrometry and its application, and to the practical man, whose chief interest is improvement and uniformity in his product. The comparative ease with which the temperature measurements were made is especially noteworthy, and shows that the newer types of optical pyrometers must be workmanlike instruments that can be depended upon even when subjected to the hard and exacting conditions found in steel works and rolling mills.

While the paper as a whole is valuable, several of the methods used and results obtained are open to criticism, and it is hoped that the following remarks will be accepted as a sincere effort to help in the study of this important subject of rail shrinkage and finishing temperatures.

In the first place, it seems to me that the relation between finishing temperature and shrinkage has been arrived at in a needlessly indirect way. The temperatures were taken directly in the mills, and it would have been very easy to obtain the exact shrinkage on the same rails that were observed. Instead of this, pieces were cut from rail flanges, welded together either by electricity or oxy-acetylene, and the coefficient of expansion obtained carefully and laboriously in the physical laboratory. Having obtained this coefficient, the finishing temperature corresponding to a given shrinkage was worked out. The results so obtained are questionable on account of the influence of the welded joints. It would not have been hard to get pieces from the mills of the required length, about 20 in., and it is to be regretted that this was not done.

According to these results there is quite an appreciable difference in the expansion of open-hearth and Bessemer rail steel, which would cause Bessemer rails to be finished about 100° F. hotter than open-hearth in order to have the same shrinkage. Regarding this they write: "This great difference in the shrinkage of the two classes of rails does not appear to have been considered in rail specifications, although this difference must have been detected in mill practice." As a matter of fact there is very little difference in the shrinkage of the two classes of steel. At our mill, Bessemer rails 100 lb. to the yard, 33 ft. long, require $6\frac{1}{2}$ in. at the hot saw, while open-hearth rails of the same section, rolled and finished at practically the same temperature, require as a rule from $6\frac{1}{2}$ to $6\frac{5}{8}$ in.

In regard to the temperature measurements one thing should be carefully noted, and that is that they were chiefly made on the head of the

rail, which is the hottest part of the rail section. The ends of the flanges are finished much colder even with the heavy-base rails, and this difference is particularly marked with thin-flange rails such as the New York Central uses. It is obvious that the finishing temperature of every part of the rail section must be well above the critical range so as to avoid brittleness, and it is certain that this will cause the rail heads to be finished a good deal hotter than is regarded as sufficient in the paper—that is, about 1,300° to 1,350° F.

The section on microstructure is very interesting, although it is stated that no definite conclusions can be drawn as to the relation between grain size and finishing temperatures. One conclusion is arrived at, however, that appears to be definitely proved from the known evidence on the subject, and which is very valuable. The paragraph may be quoted. "It would appear probable, then, that the restoration of the crystalline arrangement which is so badly shattered by the squeezing action of the rolls is not so rapid as is generally supposed, and though the restoring process begins immediately after the rolling ceases it is not complete until at a lower temperature, which in fact is the temperature which determines the grain size of the finished product."

In other words, the effective finishing temperature as it concerns the properties of the finished rail is below the temperature as observed by the pyrometer by an amount that is not thoroughly determined, but which must be allowed for. The temperature, therefore, on leaving the rolls must be sufficiently above the critical range to allow the restoring process to get well on its way in all parts of the rail section.

The paper also lays emphasis on the influence of the amount of reduction on the grain size, and the greater the reduction the smaller the grain size. This does not refer to the number of passes but to the actual reduction from the cross-section of the ingot to that of the finished rail. In this connection it does not seem as if enough importance has been accorded to one factor governing the grain size: namely, the rate of cooling. The larger the finished rail the larger the section of the various passes from the bloom to the rail, and therefore the slower the rate of cooling on the mill tables and the hot bed. To follow this further, the slower the cooling the higher the finishing temperature, and the greater the time interval between the effective finishing temperature and the critical range. This will inevitably bring about a larger grain size in the finished rail.

Finally, in regard to the difference in temperature between the inside and outside of the head of a rail, it is probable that the difference in practice is greater than that found by the writers in the laboratory. It must be remembered that the rail ingots are charged into the soaking pits while the interior is very little if at all below the freezing point, while the outside is usually below a good rolling temperature. Even after the

customary time in the pits the interior is undoubtedly hotter than the outside, and this difference is increased during rolling because of the cooling on the tables and the water used on the rolls. This difference in temperature is important because of the shrinkage stresses that are liable to be set up, and which must be carefully considered.

M. H. WICKHORST,* Chicago, Ill.—I have made a few temperature measurements, and, although Dr. Burgess has described me as an optimist on the subject, I have not felt all the confidence in the correctness of the measurements that I would have liked, and the Bureau can, I think, do considerable toward advancing the art of measuring rolling temperatures to a point where we can accept such measurements with confidence.

I wish now to make a few remarks on the subject of the relation of the rolling temperature to the general question of rail failures. Some years ago it was customary to ascribe rail failures to too high finishing temperatures of the rail, necessitated by the thin flange, but it now seems that not many railures can be traced to this cause. Very briefly I may say that a large portion of rail failures have their origin in excessive segregation, mostly of carbon and phosphorus, in the interior and upper part of the ingot. Another large portion of rail failures have their origin in seams in the bottom of the base. Excessive segregation is to be avoided by using quiet-setting steel, well deoxidized with silicon, titanium, or aluminum; in other words, ingots with "horny" tops should not be used. Of course, such quiet-setting steel pipes deeply and successful use of such steel will then, perhaps, require a large discard, or perhaps the development of a commercially available sink-head process of casting the ingot. The surface seams, of which those at the bottom of the base are especially harmful, seem to originate in the tearing of the sides of the ingot in the early stages of blooming, and probably a full study of this part of the subject can be expected to result in the elimination of a great many of the seams.

Having obtained an ingot of fairly even composition, and having given close attention to the rolling of the bloom, it would then seem to be necessary to give attention to the matter of rolling temperature in order to still further improve the properties of the rail.

Dr. Burgess found a considerable difference in the hot shrinkage between Bessemer and open-hearth rails, amounting, I believe, to something like $\frac{1}{4}$ in. difference in a standard rail length of 33 ft. My own work has indicated that the shrinkage of the hot rail increases as the carbon in the steel increases and the difference between steel of average Bessemer carbon of 0.50 per cent. and steel of average open-hearth carbon of 0.69 per cent. would amount to about $\frac{1}{4}$ in. (See *Bulletin of the American*

* Non-member.

Railway Engineering Association, vol. xvi, No. 170, pp. 163 and 164, October, 1914. It is also expected that this article will appear in the *Proceedings of the American Railway Engineering Association* for 1915.)

Dr. Burgess calls attention to the point that the shrinkage of the rail is not a correct measure of finishing temperature, and this I think we can concur in. At the finishing temperature the rail shrinks at the rate of about $\frac{1}{100}$ in. per second for a standard length of 33 ft. of cold rail, and of course the amount the rail shrinks after leaving the saws will vary according to the length of time elapsed between the finishing rolls and the saws.

As regards the question of grain size, we may ask several questions: To what extent is the grain size a function of finishing temperature; to what extent is it a function of reduction in rolling; to what extent is it a function of the length of time the ingot has been in the pit or the bloom has been in the reheating furnace; or again, to what extent is the wear of a rail a function of the grain size, or of the finishing temperature? These are some of the questions that need attention and can be finally worked out only after a considerable amount of experimental work.

JAMES ASTON,* Cincinnati, Ohio.—I have been a consistent advocate of the doctrine of the correlation of grain size and physical quality; that, other things being equal, the strength of a steel is inversely proportional to the size of grain. While it is unwise to attempt to draw conclusions from incomplete data, and in the present instance it may be somewhat unfair, I was interested in reconciling the results of the paper under discussion with the above doctrine, and was surprised and disappointed to find that the expected confirmation is lacking.

Upon arranging the data of the several tables of the paper in the order of increasing grain count, together with in each instance the carbon content, physical quality, weight of section, finishing temperature, mill practice, etc., certain conclusions seem evident. That the rails representing the materials and practice of the several mills appear to be rather promiscuously distributed, and there seems to be no regular order as to weight of section with respect to grain and temperature. And more conspicuously, that there is no concordant relation of grain size as given by the count, and the finishing temperature of rolling. Again, making due allowances for the effect of carbon, the strength does not follow the order of increasing fineness of grain; and this holds even more noticeably for the relative ductilities. Thus, of course, it is equally true that there is no correlation of physical quality and finishing temperature.

On the face of such returns we can hardly expect the mills to rush wildly into the adoption of lower temperatures of rolling; at least until

* Non-member.

more consistent and conclusive data confirming the grain-size doctrine are available.

With respect to the statement of the paper that the rolling operation produces "an 'effective finishing temperature' somewhat below the observed one" and "it would appear that the grain size of the steel is determined as much by the total amount of reduction in the rolls as it is by the finishing temperature, or any other factor," a logical answer seems to be, "Why not?" We grant that mechanical working is a factor in reduction of grain size. Consequently any reduction in the later stages of rolling, and especially that in the last or finishing pass, will break the grain down to a size less than that normal to the finishing temperature. This grain will tend to grow to normal during cooling, but will require appreciable time for its consummation; and during such interval, the temperature will be falling below that of the finishing pass. So that equilibrium will be reached with a grain growth arrested not at the finishing temperature, but at one somewhat below. The spread will be dependent primarily upon the amount of the final reductions, but to a degree also upon the speed of cooling, which in turn is influenced by weight of section, and other factors.

In view of the above, the suggestion of the authors that there may be some advantage in measuring the temperature of the blooms as they enter the rail mill, with the assumption of a constant speed of rolling, would seem open to question. For would we not have other disturbing factors varying with the amount of final reduction, weight of rail section, influence of chemical constituents, details of mill practice, and the like; all making for variations in grain size from that to be expected from temperature conditions alone?

Speaking of rail breakage, the severe winter of 1911-12 resulted in an epidemic of such failures in the Northern States. I recall an instance of the breakage by a single flat wheel of upward of 100 rails on an 85-mile stretch of as fine a piece of roadbed as there is in the country. Are we not too ready to jump at conclusions in reaching for a cause; to ascribe failure to excess of phosphorus, to incipient fissuring, to coarseness of texture resulting from high temperature of finishing, or like causes? Was not the failure in all likelihood due to the proper combination of conditions; of high phosphorus, of abnormal grain, of increased brittleness of the steel due to the low temperature, of heavy wheel load, and of excessive pounding of the flat wheel—of the above singly or in combination? And especially, was there not the important factor of unusual rigidity of frozen roadbed, which took away its power of yielding, and threw the entire resistance upon the overburdened rail?

E. F. KENNEY,* Johnstown, Pa.—I read Dr. Burgess's paper, and, by the way, I was with Dr. Burgess part of the time when he made the

* Non-member.

measurements used in this paper. The thing that impressed me first, aside from the very valuable information that is included, is that, I think, the author is reasoning a little along theoretical lines without considering some of the practical lines that would appeal to those in the rail business. Two points should be determined before we lay too much stress on this question of finishing temperatures.

In the first place, have we any real evidence that lower finishing temperatures would result in betterment to the rails? We have not seen any as yet. Some years ago we rolled some rails with widely different finishing temperatures; those rails were from the same heat and were put in on Horseshoe Curve on the Pennsylvania Railroad, and very careful records of measurements were kept showing the wear. We were not able to differentiate between the hot- and the cold-finished rails in that experiment; that is, as far as wear was concerned.

As bearing on the effect of finishing temperatures on the strength of the rails we submit some recent experience. We started to roll a rail of very heavy section, weighing 125 lb. per yard, which because of its chemistry partly, and of its section, is very stiff. The drop tests required were difficult to meet because of liability to breakage. With a view to getting greater toughness we had some of the rails rolled at a temperature 100° F. lower than our ordinary finishing temperatures, or about 1,550° F., which is rather low for rails. We did not go any lower than that, because we were afraid of breaking the rolls. These temperatures were measured at the center of the base, and the edges of the base were, of course, considerably colder. Under the drop test there did not seem to be any difference between these rails and those which were finished at the ordinary temperature. We are therefore led to believe that within the limits of temperature within which the steel can be rolled, we will not get very material difference, as far as the drop test is concerned, between hot-finished rails and cold-finished rails. But let us look at the other side of this matter.

Even if we were assured that we might get some betterment from very cold-finished rails, a number of difficulties are in the way which I think would be very hard to overcome. In some of our products we strive for a very smooth finish. This is well known over the country as "Gautier Smooth Finish," and is obtained by holding the bars in the course of the rolling, so that at the finishing pass the temperature is so low that the heavy scale does not form, and the result is a bar with very smooth, black finish. Tests on those bars show that they have a finer grain and a higher elastic limit than ordinary bars, but the ductility has been reduced in the same proportion.

Even if we could roll rails under the same conditions, I think we would lose in ductility quite as much as we would increase in other qualities. This cold rolling can be done without great difficulty on

simple sections such as rounds, squares, etc., but when it is attempted on any complex sections we almost invariably get into trouble. The bar is likely to twist, and this would be fatal in the case of rails, as they would not be accepted. Our present T rails are of very complex section as far as rolling is concerned.

Taking into account the doubt as to the practical effect of colder rolling and the difficulties attending it, I doubt very much if we are going to get much betterment in our rails, due to this colder finishing temperature.

A. W. GIBBS,* Philadelphia, Pa. (communication to the Secretary †). —The starting point or datum line of Dr. Burgess's paper is the critical temperature in the neighborhood of 700° C., 1,300° F. Running through the paper is the idea that rails should be finished at a temperature slightly above this critical temperature; notably at the bottom of p. 31,² where it says:

"From the thermal data, it follows that all these rails could have had work done on them to advantage during the rolling process down to 700° C. (1,292° F.), whereas for none of them was the actual finishing temperature lower than 880° C. (1,615° F.)."

Further on in the paper, when dealing with the question of the shrinkage clause as used in the different specifications, the excess of temperature above this critical point, as determined by the shrinkage clause together with the coefficients of expansion determined in this paper, is taken as a measure of the badness of the rolling. I would refer particularly to paragraph on page 62, reading:

"A comparison of the shrinkage clause in American rail specifications (for example that of the A. S. T. M.) with the expansion of rail steel shows that this clause permits finishing rails at 1,120° C. (2,045° F.) or 450° C. (810° F.) above the critical range of rail steel, and above the temperature at which many ingots for rails are actually rolled in practice. Such a shrinkage clause therefore does not serve the avowed purpose of limiting the finishing temperatures to a value slightly above the critical range."

While following this is the statement by the author that this is but a preliminary investigation, nevertheless I think it is unfortunate that the desirability of finishing at a point slightly above the critical range should be stated as though it were proved.

Those who are following the question of rail specifications are continually met with the statement that certain mills have rolled at finishing temperatures much below the customary ones and that the service results have not been satisfactory.

The steel people, if they will, are in the best position to offer definite proofs.

Before departing very far from the present shrinkage allowance in

* Non-member.

† Received Oct. 13, 1914.

² *Technologic Paper No. 38, U. S. Bureau of Standards.*

the rail specifications under which we are rolling, it seems necessary to have some direct evidence on the following points:

Is a greater fineness of grain than we now secure accompanied by a greater safety against shock, and against the development of that form of detail fracture concerning which so much guessing has been done of late? If we are now getting the desired fineness of grain in certain parts of the rail, such as the flange and the web, is it desirable to lower the finishing temperature still more, so that this fineness of grain will be found in the head? Will this rail of finer structure be less able to withstand the drop test? If so, are we justified in reducing the severity of the drop test, and what will be our standing in court in the event of a wreck from rail breakage, where it has been shown that the rails have been subjected to a lessened drop?

W. A. AIKEN,* New York, N. Y. (communication to the Secretary).—In a more or less detailed connection of some 15 years with the study of rail steel and the inspecting of the finished product, I have always contended, recognizing as I did the necessarily antagonistic interests involved between the producer and consumer, happily very much more marked some years ago than existent to-day, that the most important point to be definitely settled first, was the absolute necessity to secure as perfect material as possible whereon to base conclusions, arrived at from the study of its heat treatment. While it may be broadly admitted that good material may be abused during manufacturing processes and poorer material possibly improved by such processes, it is a self-evident proposition, in my opinion, that with the finished product under discussion in this paper, not to grasp the vital necessity of studying the best material obtainable is more or less a waste of time and energy and in direct opposition to the theory of the conservation of the resources.

The absolute necessity of sufficient discard from the ingot to insure sound material from which to roll the finished product has long been recognized in principle and has grown in practice; but until the two divergent interests involved become less of an *impasse* than in the past, every one is wasting energy to a certain extent in studies, no matter how elaborate, of material which the *ex parte* student must insist is not as good as can be procured. So that, while in no way disparaging the good work done in the past, as shown in the files of our various technical societies and markedly brought out by the present work of the associated physicists of the Bureau of Standards, no one interested in obtaining better rails, and confronted with the yearly reports of failures of this material, must close his eyes to further and continued effort to obtain better steel from which to manufacture rails. Meanwhile, of course, all study on the finished commercial product cannot be considered entirely wasted, but the value of heat treatment to prevent injury to good material

* Non-member.

† Received Oct. 5, 1914.

strikes me as much more important than its value in improving other material, not of the best, which is offered under present commercial conditions.

That the shrinkage clause is at best a poor makeshift, cannot be ignored. That proper control of heat treatment can be accomplished instrumentally, I think is established by the present paper; though long since abstractly recognized as feasible commercially by all investigators not biased by commercial interests.

Personally, I have had sufficient experience in the use of pyrometers to have long since felt assured that their application could be effectively made for the determination of temperatures under ordinary rail-rolling conditions. And, finally, the use of such instruments of precision will, in my opinion, settle practically once for all the fact that better finished product will be obtained at much lower finishing temperatures than are now generally used by manufacturers.

It must be recognized, however, that manufacturers cannot arbitrarily be expected to void purely commercial interests for technical ones, even though these latter may be reasonably expected, if substituted, to furnish better material. There should be no question between producer and consumer thoroughly to try out commercially any method promising as good results as the practical and positive control of heat treatment of rails.

DR. G. K. BURGESS (written communication*) I wish to express my gratification at the interest expressed in the paper by so many competent authorities. The matter that appeared to be of the greatest general interest, and which it is only too apparent is still an open question, is, What are the most suitable temperatures, *all* things considered, for rolling and finishing rails? As pointed out in several communications, the answer to that question is necessarily a complex one but it is believed to be a determinable one for any given rail section. The allied question, What is the best rail section? is intimately related to the former, as stated in several of the communications, but the answer to this appears to be even more complex. To neither of these questions does the paper give a reply, although more than one of the contributors to the discussion appears to have obtained the impression that there is here presented an argument for finishing rails at lower temperatures than is at present practiced. Emphasis has been placed, however, on the conclusion from this investigation that the present "shrinkage clause," which was designed to limit the upper temperature of finishing rails, does not accomplish this purpose. Statistics as to rejection of rails under this clause would be of interest. In this connection, the remarks of Professor Sauveur and of Mr. Trimble are of particular interest.

* Answer to discussion.

Mr. Gibbs considers "it is unfortunate that the desirability of finishing at a point slightly above the critical range should be stated as though it were proved." Of the two quotations cited by him in support of this inference, the first refers to what we consider to be a generally accepted fact, that work can be done to advantage on cooling steel to nearly the critical range; the second has to do with the inadequacy of the "shrinkage clause" to fulfill the purpose of its framers, which was to restrict the upper limit of rolling to a temperature slightly above the critical range. Whether or not rails should be rolled colder or hotter than is the present practice it was not intended to advocate in the paper, and, as Mr. Gibbs states, "the steel people, if they will, are in the best position to offer definite proofs" of the qualities of rails rolled hot or cold.

The fundamental questions asked by Mr. Gibbs regarding the relation of fineness and uniformity of grain, finishing temperatures, resistance to shock, and breakage in the track, can only be answered by experiments on an elaborate scale; and there is also implied here the underlying questions of weight and section of rail.

The comprehensive communication from Mr. Webster is most welcome and timely, and we are glad to note and acknowledge his valuable contributions to our knowledge of the properties of rails and in particular his urging repeatedly during the past 25 years the systematic study on an adequate basis of the fundamental problems of the physics of steel.

It is of more than retrospective interest to look back at the Suggested Lines for the Discussion and Investigation of the Physics of Steel inaugurated at the Chicago meeting of this Institute in 1893, and to note how many of the queries then propounded still remain unanswered, and this in spite of the nearly boundless facilities of our many iron and steel manufacturing establishments, and the existence of not a few laboratories. One cannot help but feel that many of these questions—some of which Mr. Webster repeats to-day—might have been answered ere this without sacrificing the efficiency of our mills and probably with great economic gain in many instances.

That the government standardizing laboratory is interesting itself—within its necessarily limited sphere of activity—in some of the fundamental problems of the properties of railway material should, we believe, serve as an added stimulus, however slight, to others in the common endeavor to solve these oftentimes perplexing, and happily sometimes easy problems, but ones the solutions of which have been deferred.

Mr. Webster as early as 1890 to 1893 reached the conclusion from his investigations on The Relation between the Chemical and Physical Properties of Steels "that the control of the rolling temperatures was the most important factor;" and yet, to-day, it would appear that neither he nor any one else can say with certainty at what temperatures rails should be rolled.

Regarding the difficulties to be encountered, some of which Mr. Webster mentions, in the practical solution of this problem, we believe some of them to be less serious than might at first appear. There certainly will be no difficulty, for instance, in identifying individual rails the temperatures of which have been measured. Mr. Webster mentions the procedure, under the shrinkage clause, for the inspector to follow in a mill with four saws; but how about the mill with one saw? This mill also works continuously, but has no automatic shrinkage control.

Again, the question of rail section vs. temperature distribution, although undoubtedly troublesome for rails of thin flange, may perhaps be answered; and we hope to be able to contribute to its solution.

To Mr. Webster's suggestions for more work along more extended lines, for taking into account all the factors entering into the problem of rail temperatures and for co-operation with the practical mill men, we most heartily assent.

To his questions as to effect of section and the practical possibility of rolling rails colder than is the present practice, we must plead ignorance.

In view of the fact that the shrinkage measurements made in the mill and the expansion measurements made in the laboratory agree very well for the few examples given in the paper for open-hearth rails, we are inclined to the belief that these measurements are trustworthy; but much more data on shrinkage and finishing temperatures should be obtained for different kinds of steel and for rails embracing at least all the usual sections.

The communication by Mr. Waterhouse contains many useful suggestions which must be kept in mind in a comprehensive study of the subject of rolling temperatures vs. properties of rails.

Regarding his criticism of our method of determining expansion in the laboratory rather than of shrinkage in the mill, it may be stated that our original program as planned, and as carried out in the mills, involved no interference with mill practice. He appears to overlook the fact that shrinkage measurements were made, however, in one of the mills as shown on p. 55 of *Technologic Paper No. 38*. The results in shrinkage agreed with those on expansion. The effect of welding the rails can easily be shown to be *nil* as carried out. Mr. Waterhouse points out the importance of the metallographic examination indicating an effective finishing temperature lower than that given by the pyrometer. We believe this to be a field worthy of further investigation.

Mr. Aiken, in the light of his considerable experience, insists on the desirability of improving rail material, with which every one would probably agree; but we also believe in the careful study of the manufacturing processes and properties of the resulting products as carried out under present commercial and technical conditions with the materials in general use.

Mr. Wickhorst points out in a comprehensive manner what he considers to be the sequence of importance in the causes of rail failure, and emphasizes the uncertainty in practice of the effect of finishing temperatures and the need of more experimentation on this and similar questions, with all of which we agree.

Mr. Kenney's statement that 100° F. either way from the "ordinary" finishing temperature makes no material difference in the properties of rails of heavy section is of importance, as well as his account of rails from "the same heat" but rolled at "widely different" finishing temperatures not showing appreciable differences in the track. The detailed and complete account of this, and similar experiments on a more exactly determined basis, involving a sufficient quantity of material to yield "practical" results, would be of interest.

Until this is done, the question of suitable finishing temperatures will remain unsolved, for the "theoretical" man still asks, Why not roll colder? and the "practical" man admits he has no certain evidence one way or the other.

Mr. Dudley has given us in his elaborate communication what is practically a new paper on the subject of the service of rails as influenced by manufacturing, maintenance, and weather conditions. His explanation of the abnormal number of failures for the year ending Oct. 31, 1912, as due to the cold winter is undoubtedly correct. The experience of the New York Central Lines that the Bessemer rails have an overwhelming proportion of failures as compared with open-hearth—or 22 to 1—appears not to be borne out generally. Mr. Dudley does not give, however, the total number of Bessemer rails, so that a strict comparison cannot be made between the failures of the two types from his data on the New York Central Lines. For the year 1912, for the whole country, the ratio of Bessemer to open-hearth rail failures per million tons laid was 5,160 to 2,379—or roughly 2 to 1. It would appear from Mr. Dudley's figures that the present breakage of open-hearth rails on the New York Central Lines is 91 per year per million tons laid, or only about $\frac{1}{26}$ the average for the whole country in 1912; from which one would infer that the breakage is in general unnecessarily high, and one should perhaps be cautious in attributing such wholesale failures too readily to increased train loads and high speeds. The questions of maintenance of track and quality of the rails and rolling equipment are perhaps of at least equal importance—factors which Mr. Dudley emphasizes.

Regarding the question of best rolling temperatures, Mr. Dudley appears to consider the unsatisfactory experience with the Kennedy-Morrison process of holding the rail bar 40 sec. before the last pass, to have demonstrated the practical unsuitableness of colder rolling. The fallacy involved in that pseudo-cold-rolling process was explained at the time of its introduction by Professor Sauveur. Mr. Dudley's conclusion

concerning shrinkage limits prescribed by the school favoring cold rolling loses some of its force in view of the fact that the shrinkage limits have almost without exception permitted rolling at very much higher temperatures than has apparently been generally supposed.

The question of transverse fissures, also raised by Mr. Dudley, we agree is one requiring further study for their explanation and elimination. The Bureau of Standards has begun an experimental investigation of this subject—following in part suggestions from Mr. Dudley.

Manganese Steel and the Allotropic Theory

Discussion of the paper of ALBERT SAUVEUR, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2439 to 2449

G. K. BURGESS, Washington, D. C.—This very instructive contribution by Professor Sauveur to our knowledge of that puzzling substance, Hadfield's manganese steel, should not only be read in connection with the paper of Messrs. Hopkinson and Hadfield referred to by Professor Sauveur (*Bulletin* No. 87, pp. 513 to 530, March, 1914); but one should also have in mind, as throwing much additional light upon the questions raised, two papers recently presented before the Iron and Steel Institute, the one by Sir Robert Hadfield (*Journal of the Iron and Steel Institute* No. II, 1913) on Heating and Cooling Curves of Manganese Steel, the other by Messrs. Hadfield and Hopkinson (*ibid.*, No. I, 1914) treating of the Magnetic and Mechanical Properties of Manganese Steel. We then have brought together for this substance, the thermal, magnetic, and mechanical properties as well as the resulting microstructure, for the definite heat treatments described, and in particular after long annealing between 500° and 700° C.

Sir Robert Hadfield and Professor Hopkinson see in the properties found for manganese steel (of C = 1.25 and Mn = 12.5 per cent.) a demonstration of the untenableness of the allotropic theory of iron, whereas Professor Sauveur reaches the contrary conclusion, that some of the facts observed can be explained only in terms of the existence of beta iron in steel of this composition.

There is another possible hypothesis: namely, that these several series of experiments shed no light on the subject of the allotropy of iron, or at least as regards the existence of beta iron. However heretical it may sound, I am inclined to this last belief.

For steel with 1.25 per cent. carbon, we are in that portion of the Fe-C diagram, in any of its forms usually given, in which, for equilibrium conditions, or very slow heating and cooling, there is no beta iron possible, unless the improbable assumption is made that the carbon has combined mainly with the manganese rather than with the iron. But it is pre-

cisely when the equilibrium conditions are approached in heating and cooling that those properties of manganese steel are found on which are based the arguments of Messrs. Hadfield and Hopkinson on the one hand and of Professor Sauveur on the other. It would appear that if any beta iron were to be formed in heat treating this steel, it would be expected to be found only after relatively rapid cooling due to the non-homogeneity of the mass; but my understanding of the claims of Professor Sauveur is to the effect that beta iron is supposed to be liberated on long annealing (or tempering) between 500° and 700° C. after rapid cooling, but to be practically non-existent for rapid cooling and heating.

It would appear reasonable to attribute to the 12.5 per cent. of manganese here present—in its relation to the iron and carbon—the major rôle in the phenomena observed, although the effect of high carbon with little or no manganese in producing somewhat similar but less intense effects is not to be neglected, as pointed out by Messrs. Hadfield and Hopkinson.

The fact that the thermal and magnetic A_{123} point of this manganese steel is lowered from some 700° C.—evidently by the manganese—to liquid-air temperatures, or even suppressed, for ordinary rates of cooling, and is recovered, as shown by Messrs. Hadfield and Hopkinson, by long heating between 500° and 700° to nearly the normal value for carbon steel, or to 674° C., would seem to indicate that it is the manganese which is mainly responsible, this element acting in its well-known deterrent capacity and perhaps otherwise also.

It would be of interest to take (which does not seem to have been done) the cooling curve of this steel immediately after long annealing at 500° to 700° C.; the A_{123} point should then be found as sharply marked somewhere between 670° and ordinary temperatures as is the Ac_{123} point at 674° C., found by Messrs. Hadfield and Hopkinson. That the A_{123} transformation must have taken place is evident, since the steel is magnetic when cold after annealing at 500° to 700°; and the transformation from the non-magnetic to the magnetic state certainly involves the same quantity of heat and is probably as sharply located as the reverse phenomenon, although perhaps not at the same temperature, as is found for example in some nickel-chrome steels, for which A_{123} may be some 400° C. below Ac_{123} .

No argument is here offered for or against the allotropic theory of iron, but it is merely desired to point out why the existence or non-existence of beta iron would appear to play no rôle in defining the properties of this manganese steel, when equilibrium conditions of heating and cooling are approached.

DR. J. E. STEAD, Middlesbrough, England (communication to the Secretary*).—In discussing the allotropic theory I think that every

* Received Dec. 7, 1914.

metallurgist of repute agrees that there are two allotropic modifications of iron, the alpha and the gamma. The question always in doubt, and never fully answered by Osmond, is whether there is a third or a beta modification. The strong magnetic properties of hardened carbon steels always have been a stumbling block in the way of the beta advocates.

I have regarded with the greatest respect the experimental work of Osmond and his reasoning, which always were of the highest order, but after all that has been said for the beta theory one could only accept it tentatively as a reasonable hypothesis.

The peculiar properties of manganese steel depend on the carbon. Take carbon away, and, no matter what the heat treatment may be, it always remains magnetic. Manganese may be reduced to 2 per cent., but if the carbon is raised to 1.8 per cent., one obtains material which has some of the remarkable properties of manganese steel containing 12 per cent. manganese and 1.2 per cent. carbon.

I agree with the author that the magnetic properties in steel are coincident with the presence of alpha iron.

Austenite in cold steel has no martensite structure, but this structure, as the author has said, can be produced by annealing austenite. The coarseness of that structure depends on the temperature and time of heating before quenching. It is fine if the temperature has been high and of short duration; coarse if the preheating has been high and prolonged.

The structure developed by annealing austenite at about 600° C. is evidence of the crystalline structure of the austenite, which, being homogeneous, is not revealed by the ordinary etching reagents. Annealed austenite and its equivalent in quenched pure carbon steel have the martensite structure readily revealed by etching, one part along the cleavages being more readily attacked than other juxtaposed portions—a proof to my mind of non-homogeneity, a difference from austenite in possibly both chemical and physical constitution.

Very coarse martensite can readily be split along its cleavage planes, and in my opinion these same cleavage planes pre-existed in the metal when in the pure austenite state, but in that state could not be readily separated owing to the ductility of the whole mass. If we accept this basis that martensite is not homogeneous, what then is the partial change that occurs in austenite which is coincident with extreme hardness? Only one thing is certain, that one part of the martensite is magnetic, and therefore contains alpha iron; it may also contain, and probably does contain, some austenite.

The author, while concluding that the hardness of quenched and reheated manganese steel is due to the formation of a considerable proportion of beta iron, does not, I think, bring any proof excepting that of extreme hardness to show that any beta iron is really present. Is it not possible that, if there is any truth at all in the intermolecular pressure

theory, the expansion which occurs when a portion of the gamma iron changes to alpha, may be responsible for the hardness? It would, of course, be impossible by the microscope or any available means to discover the alpha iron, or the intermolecular pressure, but it seems quite as reasonable, if not more so, to assume that the hardness is due to interstress between alpha and gamma molecules rather than to beta iron + gamma iron + alpha iron.

I quite agree with the author that all hypotheses should be regarded with respect, and none of them should be dismissed without most careful consideration. I would also point out that nobody at present has sufficient data to justify any dogmatic conclusion being formed. What we want is more data, more research and more experimental evidence.

The Surface Decarbonization of Tool Steel

Discussion of the paper of J. V. EMMONS, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2233 to 2248.

ALBERT SAUVEUR, Cambridge, Mass.—Mr. Emmons is to be congratulated for having placed before us in so forcible and satisfactory a manner a question of much importance—that of the surface decarbonization of steel. Of course, both manufacturers and consumers have known for many years that during the heat treatment of high-carbon steel, surface decarbonization is always likely to occur to a greater or less degree, but the subject does not seem to have been investigated with the method and thoroughness it deserves.

I am in substantial agreement with Mr. Emmons's conclusions, and I have but little to add to what he has so well said. His statement that carbon dioxide is just as effective an oxidizing agent as pure oxygen is likely to cause surprise and some will wonder whether his investigation has been carried far enough to warrant so unexpected a conclusion. The reaction $C + O = CO$, expressing the oxidation of carbon by pure oxygen, is decidedly exothermic, whereas the reaction $C + CO_2 = 2CO$, expressing the oxidation of carbon by carbon dioxide, is endothermic. Decarburization by pure oxygen, therefore, should take place much more readily than by carbon dioxide.

One cannot help regretting that the author does not mention the best way of preventing in a practical manner surface decarbonization while treating tool steel and other high-carbon steels.

BRADLEY STOUGHTON, New York, N. Y.—In connection with the very interesting paper which Mr. Emmons has presented, reference should be had to a paper¹ bearing in an interesting way on the results

¹ The Decarburisation of Steels in the Salt Baths used for Heating Prior to Hardening.

that Mr. Emmons obtained in decarbonization, which was prepared by A. M. Portevin for presentation at the meeting of the Iron and Steel Institute of London in September, 1914.

E. G. SPILSBURY, New York, N. Y.—The point brought out by Mr. Emmons regarding the activity of carbon dioxide as an oxidizing agent, has been fully recognized in the manufacture and distillation of zinc from its ores. In the presence of carbon dioxide, metal in its nascent condition is very rapidly decomposed.

One of the great difficulties in the distillation of zinc arises from the great affinity of the metallic vapor for oxygen, and the aim in the whole process is to exclude the presence of oxygen as much as is possible. This is why so far we have not been able up to the present time to improve on the use of comparatively small units of distilling retorts.

Even then, from the process itself, carbon dioxide in considerable quantities is formed, and at the temperature inside the retort would very actively react on the metal vapor. It is therefore necessary to pass the vapor as rapidly as formed into a zone of temperature below the point at which the carbon dioxide becomes inert. In this case the reduction of temperature from the point of volatilization of $1,100^{\circ}$ to 700° is necessary. This may have some bearing on the question Mr. Emmons brings up.

OLIVER W. STOREY,* Pittsburgh, Pa. (communication to the Secretary†).—While carrying on researches in the annealing process for malleable castings the writer became interested in the surface effects produced during the annealing of white iron. It was noticed that various commercial malleable products differed, not in the constitution of the "black heart," but in the surface layers. Some products showed a skin consisting entirely of pearlite; others a skin of two layers, the outer of ferrite, beneath which was one of pearlite; while some had a skin consisting entirely of ferrite.

In the investigation a large number of packing materials were used for the annealing process. These included oxidizing materials, neutrals, and materials evolving carbon monoxide.

The results showed that the chemical nature of the packing material was usually of little importance, with a few exceptions. The method of packing the castings was of greatest importance. This influenced the circulation of the furnace gases in the annealing boxes, since closely packed boxes allowed less circulation than those loosely packed. Therefore, the composition of the gases in the annealing furnaces was also important. It was found that in those boxes which were loosely packed, readily allowing circulation of air (an electric ribbon-wound furnace being used as the heating medium), the resulting malleable iron had a skin which consisted almost entirely of pure ferrite, with little pearlite, due

* Non-member.

† Received Oct. 15, 1914.

to the oxidation of the surface carbon by the air. Where the packing material consisted of a closely packed mass such as finely ground fire clay the skin consisted of ferrite and pearlite, since the oxidation was retarded. Where ground carbon was used the surface usually consisted of pearlite, though here the temperature of annealing was important.

When a highly oxidizing packing was used all of the carbon was removed from the surface. If a case-carbonizing mixture at a high annealing heat (900° C.) was used the surface of the malleable iron consisted of a high-carbon steel, giving a hard surface. When carbon was used as a packing material the temperature was important owing to the equilibrium conditions of the reversible reaction: $\text{CO}_2 + \text{C} \rightleftharpoons 2\text{CO}$.

At the lower temperatures CO_2 is in excess and castings annealed at these temperatures had a decarbonized surface though packed in carbon. This was due to the action of CO_2 , as pointed out by Mr. Emmons in his paper. At the higher temperatures more CO was present and the surface consisted of pearlite.

From the results it will be seen that in the malleable process the composition of the furnace gases and the method of packing influence the skin structure.

In a paper before the September meeting of the American Foundrymen's Association the writer called attention to the surface action of both CO and CO_2 and the commercial advantages in being able to obtain a soft carbonless rim or one of high-carbon steel of excellent wearing qualities. The results obtained in the researches of Mr. Emmons, with slightly modified conditions, are directly applicable to the making of malleable iron.

H. O. HOFMAN, Boston, Mass. (communication to the Secretary*).—The explanation for the surface decarbonization of tool steel by J. V. Emmons is based in part on the experiments of F. Wüst (*Metallurgie*, vol. v, No. 2 (1908)) which show that *in an evacuated tube* ferric oxide gives off some of its oxygen at 600° C. and continues to do this as the temperature is raised. Thus, at

1,050° C., Fe_2O_3 is changed into $\text{Fe}_2\text{O}_3 \cdot \text{FeO}$ (= Fe_3O_4).

1,100° C., Fe_3O_4 is changed into $\text{Fe}_2\text{O}_3 \cdot 2\text{FeO}$ (= Fe_4O_5).

1,200° C., Fe_4O_5 is changed into $\text{Fe}_2\text{O}_3 \cdot 3\text{FeO}$ (= Fe_5O_6).

It has been proved experimentally by three investigations that *at atmospheric pressure* ferric oxide is not decomposed by heat alone at 1,300° C.

K. Honda and T. Sone (*Science Reports of the Tohoku Imperial University*, Sendai, Japan, vol. iii, p. 223 (1914)) found that hematite remained unchanged when heated to 1,300° C., in a current of air.

W. Mostowitsch and H. O. Hofman (*Trans.* xl, 807 (1909)) at an

* Received Dec. 21, 1914.

earlier date heated ferric oxide in a current of air to 1,500° C. and found no change in weight.

E. G. Kohlmeier (*Metall und Erz*, vol. x, pp. 447 to 483 (1913)) found that in a current of oxygen at 920° to 950° C. the red Fe_2O_3 became black, lost oxygen as the temperature was raised, and fritted at 1,370° with the formation of $\text{Fe}_2\text{O}_3\cdot\text{FeO}$.

The new compound gave up more oxygen with increase of temperature, fused at 1,470°, and formed $5\text{Fe}_2\text{O}_3\cdot 3\text{FeO}$; this further lost oxygen up to 1,520°, when there was formed $4\text{Fe}_2\text{O}_3\cdot 3\text{Fe}_2\text{O}_3$; and the last gave up oxygen as the temperature was raised, and fused at 1,600° with the formation of $\text{Fe}_2\text{O}_3\cdot\text{FeO}$, which corresponds to the well-known Fe_3O_4 .

J. V. EMMONS (communication to the Secretary*).—The writer is deeply indebted to Professor Sauveur for the favorable comments on his paper. The fact that carbon dioxide is as powerful an oxidizing agent as pure oxygen was at first surprising, although it is well known that at temperatures above the critical points of high-carbon steel it is slightly dissociated into $\text{CO} + \text{O}$. It was shown by the several experiments with different oxidizing agents that the rate of decarbonization was similar in all cases, and that in the presence of any oxidizing agent the rate of decarbonization was a function of the temperature only. This was taken as an indication that the rate of decarbonization was dependent upon the rate of diffusion of Fe_3C through gamma iron. Further confirmation of the activity of carbon dioxide as an oxidizing agent at high temperatures is found in the discussion by Mr. Spilsbury and Mr. Storey.

The prevention of decarbonization is obviously a question of preventing the access of oxidizing agents to the steel while heated above its critical points. The writer regrets that no simple rule for doing this can be laid down. The heat treatment of tool steel assumes such a variety of forms that the accomplishment of this end is largely a matter of ingenuity in dealing with specific problems.

The paper of Mr. Portevin is especially interesting for the information given in regard to the equilibrium between carbonization and decarbonization. His results indicate in a striking manner that the rate of carbonization as well as decarbonization is a function of the rate of diffusion of the carbide through the solid solution of Fe_3C in gamma iron.

There is no question that a knowledge of the mechanism of decarbonization is of great value in the production of malleable iron, as Mr. Storey so well says. The commercial possibilities of a skillful regulation of the condition of the carbon in the skin or rim of the casting are very great.

The writer will not dispute the contention of Mr. Hofman that ferric

* Received Mar. 10, 1915.

oxide is not decomposed by heat alone at atmospheric pressure, as in all the experiments performed there was present carbon monoxide, which would serve to start the decomposition. Indeed, in the commercial heat treatment of tool steel carbon monoxide is almost invariably present. Further confirmation of this decomposition of ferric oxide will be found in the work of C. M. Johnson. (*Journal of Industrial and Engineering Chemistry*, vol. i, No. 7, p. 459 (July, 1909)).

Turbo Blowers for Blast-Furnace Blowing

Discussion of the paper of RICHARD H. RICE, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 89, May, 1914, pp. 721 to 743.

S. G. VALENTINE, Oxford, N. J.—In a paper read at the New York meeting of the Institute last February I presented some notes taken from our experience with the turbo blower operating on a blast furnace. In connection with Mr. Rice's very interesting paper there are a few items that might be of further practical interest.

The question of the reliability of this type of blower will naturally be presented. In this connection we would say that our turbo blower has been in practically continuous use during the present blast, which was begun June 7, 1912, with only three interruptions for repairs of any consequence. Soon after blowing in there was developed a tendency of the blower to surge or pulsate. This occurred on operating at a point considerably below the normal capacity of the machine, and was checked with comparatively little trouble by throttling the intake slightly and taking up a small lost motion on the governor connections, a legacy of the previous blast that had not been attended to. All this required only a few hours' time. In March, 1914, the furnace being banked on account of a blizzard that cut off our freight deliveries for five days, the opportunity was used to replace the carbon packing on the steam-turbine shaft and to repack the steam inlet valves. In June, 1914, the blower was off 8 hr. to replace an oil ring and scrape a bearing. With these exceptions we have made continuous use of the machine and find we can depend upon it with entire safety as a piece of machinery.

It is reliable also in regard to its air deliveries. Being the first one built this blower was designed to throw a larger volume of air than we are able to use and is therefore run at below its normal rating, and at our running speeds the actual quantity of air furnished is probably a trifle more than the calibration would indicate. This, in our case, might amount to 1 or 2 per cent. However, the regularity of the amount delivered is evidenced by the general regularity of the furnace operation, and we find we can rely on prompt and automatic adjustments to pressure variations, maintaining thereby a constant volume. It is

rather difficult to show either graphically or in figures the element of furnace regularity; and, of course, in addition to variable blast volume there are a number of other things that might affect the regular working of the furnace. Eliminating these items, however, there is evidence in our daily experience of very regular driving and freedom from any tendency to slip or hang, as well as regularity of quality in our product, for which we are inclined to give very much credit to regularity of blast used. When running on basic iron it is not unusual for the furnace to make 90 to 93 per cent. of standard basic on a magnetite mixture, 75 per cent. of which carries sulphur to the extent of 0.50 to 0.70 per cent., which is quite satisfactory.

While not able to present any formal statements as to cost of operating and maintenance as compared with the other reciprocating steam or gas engines the figures for our own case may be of interest. For all repairs, including packings of all sorts, all pipe fittings, etc., used directly and indirectly during the last 28 months, there has been expended \$227.15. For oil, waste, and all similar supplies the amount for the same period has been \$159.22, or for both above items slightly more than two mills per ton of pig iron produced.

Mr. Rice's paper includes a very interesting description of the tests made to determine the difference in the vibration of the air deliveries from rotary and reciprocating engines. The results obtained by him seem to bear out what our experience would have led us to expect in this particular. I am personally convinced that a continuous stream of air is better than a pulsating stream of air as far as the smoothness and regularity of furnace operating is concerned. It would be perhaps difficult to specify any particulars observable in regard to this item. But certainly whatever difference may be shown to exist as between these two types of air-current deliveries to the tuyères would be in favor of the most regular delivery obtainable, which seems to be secured in the highest degree from a rotary engine.

W. MCA. JOHNSON, New York, N. Y.—The question of turbo blowers vs. piston blowers can be considered, in my judgment, entirely in the light of engineering economics. This gives the verdict to the turbo blower entirely on the capital cost, along the following lines of reasoning:

We can admit that for pressures of 25 lb. per square inch the piston blower has a somewhat better volume efficiency, say 85 per cent. as against 75 per cent.; better mechanical efficiency; and makes less flue dust in smelting than the turbo blower. But the capital cost of the turbo blower must sooner or later prove to be one-third to one-half that of the piston blower, remembering that modern high-speed turbo generators sell in large sizes for as low as \$10 per kv.a., or even lower, as against a far larger sum per unit for the slower-speed reciprocating equivalent.

It is reasonable then to suppose that eventually turbo blowers will

sell for "so much per pound." Then a great deal of $\frac{1}{2}$ mv* is raced through a few tons of iron, steel, and brass in the case of the turbo blower. Now when new construction of blast furnaces does come, as come it will, the turbo blower will appeal to bankers and to such engineers as partake of their views, for reason of its low capital cost as compared with that of the reciprocating blower.

Air is cheap and we can waste it. So is power and we can waste it. But capital is scarce and we must conserve it. The advance in the state of art is now so rapid that "commercial obsolescence" demands high charges for depreciation in metallurgical industry. It is good finance that places this at a minimum of 15 per cent. per annum. To my mind, this consideration is all important in making a comparison of the future of the turbo blower. The commercial history of the centrifugal vs. the reciprocating pump has shown the same to be true in the past.

L. IVERSEN,* Pittsburgh, Pa.—I would like to give you a few figures showing a comparison between the turbo blower and the modern reciprocating steam blowing engine of the latest type with blowing cylinders equipped with automatic plate valves, and I will place the steam-engine driven reciprocating blower on exactly the same basis as the turbo blower described by Mr. Rice, both in respect to the amount of air delivered and the steam and vacuum conditions.

I will not discuss the question of how much air is required to burn a pound of coke in the blast furnace, but I believe I am safe in saying that it will require the same amount of air for equal conditions in the furnace, whether same is supplied through a reciprocating or a rotary machine. I will base my figures on 65 cu. ft. of air per pound of coke, or 45,500 cu. ft. per minute for the same size furnace as discussed by Mr. Rice, which will give us a theoretical air horsepower of:

$$\frac{45,500 \times 5}{100} = 2,275$$

The modern blowing engine *cylinder* will give a compression efficiency of 95 per cent. and a mechanical efficiency of 97 per cent., which would give a horsepower at the piston rod of:

$$\frac{2,275}{0.95 \times 0.97} = 2,470 \text{ hp. for the engine,}$$

against a shaft horsepower of:

$$\frac{2,275}{0.70} = 3,250 \text{ hp. for the turbo blower.}$$

An indicator card taken from a blowing cylinder of this kind is shown herewith, Fig. 10. This card was taken with a five-scale spring and in the

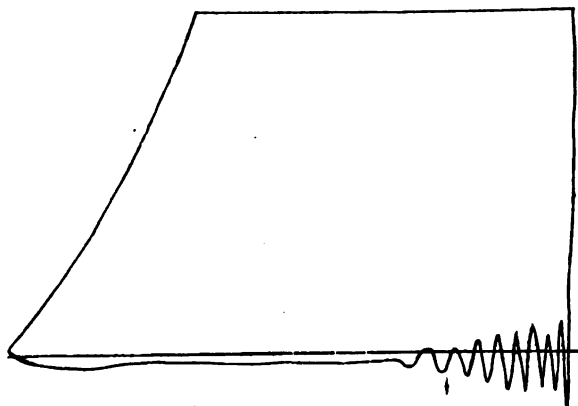
* Non-member.

most careful manner. It will be noted that the suction line rises above the atmospheric line at the end of the stroke. This card shows a volumetric efficiency of greater than 100 per cent. and the maximum suction below the atmosphere is 0.18. After deductions have been made for rise in temperature and valve resistance, we get from measurements a compression efficiency of 95 per cent.

If we substitute 3,250 shaft horsepower for the turbo blower instead of 2,500 and accept Mr. Rice's figures of 404,000 cu. ft. of gas per hour for 2,500 hp., we get a gas consumption for the turbo blower of:

$$\frac{404,000 \times 3,250}{2,500} = 525,000 \text{ cu. ft. per hour.}$$

In order to utilize the available energy assumed for operating the turbo blower to advantage in the steam engine, namely, 200 lb. pressure, 200° superheat, and 28½ in. vacuum, we will install cross-compound engines and operate same non-condensing with a low-pressure turbo



Crank end of cylinder, low-pressure air side. Cylinder 84 in. diameter; 60 in. stroke; 57 revolutions; boiler pressure 130 lb.; vacuum 28 in.; blast 20 lb.

FIG. 10.—INDICATOR CARD FROM RECIPROCATING BLOWING ENGINE.

generator which will take the exhaust steam from the engines and readily get the benefit of the 28½ in. of vacuum.

The steam consumption of this type of engine per indicated horsepower per hour will be 17 lb., the mechanical efficiency 88 per cent., and the steam consumption per hour:

$$\frac{2,470 \times 17}{0.88} = 48,000 \text{ lb.}$$

The steam consumption of the low-pressure turbine will be 27½ lb. per kilowatt and the kilowatt output:

$$\frac{48,000}{27.5} = 1,750$$

Assuming 25,200 B.t.u. per kilowatt (which I believe is far too low) and 103 B.t.u. per cubic foot of gas, we get:

$$\frac{1,750 \times 25,200}{103} = 430,000 \text{ cu. ft. of gas, which amount is to be credited}$$
 to the total amount consumed by the steam-engine driven reciprocating blower. This is using:

$$\frac{48,000 \times 1,247.4}{103 \times 0.70} = 830,000 \text{ cu. ft. per hour.}$$

Deducting 430,000 which we are getting back in the kilowatt output, we get a gas consumption chargeable to the steam-engine driven reciprocating blower of 400,000 cu. ft. per hour, which is equivalent to producing an indicated horsepower in the engine on less than $8\frac{1}{2}$ lb. of steam per hour considering the combined installation.

From the above it will be seen that the turbo blower uses 125,000 cu. ft. of gas, or 32 per cent., more than that required to operate the reciprocating blower, and since the cost of installation of the steam-engine driven reciprocating blower and the turbo blower will be approximately the same, it is evident that the turbo blower cannot compete with the former for blast-furnace work, no matter how well the structural difficulties have been overcome.

You will note the tremendous difference in the results obtained by me and those given by Mr. Rice, and since the point at issue is of great importance, I would suggest that a commission of disinterested parties be appointed to observe the results, witness and make tests in order to determine whose assertions are correct. As for myself, I cannot accept Mr. Rice's methods of measuring the air to the turbine and by means of the theoretical horsepower required to compress this air and the efficiency of the machine, to get the steam consumption. I would measure the steam going to the turbine, or possibly measure both the steam and air; the one method would then check the other. I have reasons to believe that if such a commission were appointed, they would not only discover the fact of the poor efficiency of the turbo blower, but also that the increased furnace efficiency expected from the turbo blower, due to steady blast, does not materialize and that the furnace works better with reciprocating blowing engines.

The reason for this is very easily explained when you consider that the air pressure is governed by the furnace. The amount of air delivered through the reciprocating blower will have to go through the furnace, no matter how much the pressure increases. With the turbo blower, however, it is different; its speed must correspond to the pressure against which it has to deliver. If the pressure tends to increase, due to the furnace, the turbo blower must speed up or it will churn part of the air. While the speed of the turbine can be regulated from the pressure, never-

theless there must be an increase in pressure first before there can be any force to act on the regulating mechanism of the turbine. The regulator cannot be made to act on too small pressure differences or it will cause a surging of pressure exactly the same as an engine governor that "hunts." If the regulation is effected by the volume, the effect will be exactly the same as above for the reason that a reduction in volume at a certain speed of the blower could only result from an increased delivery pressure.

J. E. JOHNSON, JR., New York, N. Y.—Referring first to the question of the delivery of the turbo blower, at the February meeting of the Institute Mr. Valentine gave the results which he had obtained by the use of one of these blowers and claimed that his furnace had only required with its use 41.5 cu. ft. of air to burn 1 lb. of coke. I criticized this statement at the meeting as being less than practice indicated as at all probable. Professor Richards censured me for making this criticism, but when I afterward intimated how strong were the grounds on which I made it, he calculated from Mr. Valentine's figures the amount of air which must have been blown, using the weights of materials charged and the gas analysis, and wrote me that I was entirely correct and that Mr. Valentine must have been using 15 per cent. more wind than the amount he stated, which was taken from the indicator of the turbo blower. This seems to me to show conclusively, especially when taken in conjunction with other data along the same line, that the calibration of this turbo blower was in error by a large percentage and that it was delivering considerably more wind than it took credit for. This error follows through all Mr. Rice's calculations. He has assumed that it takes 25 or 30 per cent. more air with the reciprocating engine, measured by piston displacement, than with the turbo blower and he has based all his figures for power consumed in the two cases upon this assumption. He is probably 15 per cent. too liberal to the turbo blower and not liberal enough by 5 or 10 per cent., on the basis of volumetric efficiency, to the reciprocating engine.

In regard to the steam engine, it seems to have been forgotten that in the country of the blind the one-eyed man is king, and that the turbo blower with its economical steam end may easily distance old, badly designed and perhaps worn-out blowing engines, and yet not equal modern ones in good condition. I, myself, have run an old engine at 19 revolutions with its outlet valve tightly shut, and there is no doubt that a turbo blower would beat this performance, particularly as its steam consumption per indicated horsepower hour was not less than 50 lb.; but the steam engine has not stood still and a reciprocating blowing engine of good, modern design, working under the conditions specified by Mr. Rice for the turbo-blower, 200 lb. steam, 200° superheat, 28½ in.

vacuum, of modern size and at a reasonable piston speed, will deliver an indicated horsepower for from $8\frac{1}{2}$ to $9\frac{1}{2}$ lb. of steam. The mechanical efficiency of a reciprocating compressor is probably higher than that of any other reciprocating apparatus because of the way in which the inertia of the reciprocating parts can be used to balance the changes in pressure in the steam and air ends. Ninety-five per cent. mechanical efficiency has been done by such engines and I have in my possession cards from one such engine showing 92 per cent. The volumetric efficiency is about 96 per cent. and the diagram efficiency about as much. The net result of these figures is that an air horsepower can be delivered by such an engine with about 10.4 to 11 lb. of steam, whereas the turbo blower on Mr. Rice's own showing requires 13.35 lb. The much smaller amount of power required, some 2,100 hp. as against the 2,500 required by the turbo blower, offsets the slightly higher cost of the reciprocating engine and gives a plant complete for just about the same expenditure as is required for the turbo blower, leaving the much smaller steam consumption of the reciprocating engine as a net advantage. In addition to the smaller boiler power required for the reciprocating plant, there is the further advantage of the very much smaller condenser, since the engine will get along almost as well on 26 in. vacuum as on $28\frac{1}{2}$, and only requires a condenser one-third as large and about one-third as much condensing water to produce it. On the other hand, it is a matter of common knowledge that the turbine loses efficiency very rapidly as the vacuum falls off. It is entirely impossible in three out of the four iron-producing districts in the country to obtain condensing water which will give a vacuum of $28\frac{1}{2}$ in. for any considerable proportion of the year and as a result the possible economy of the turbo blower must be materially reduced.

Moreover, I am advised that a test has been made at the turbo-blower plant of the Iroquois Iron Co. and that this machine showed an efficiency of 55.6 per cent. against the 70 per cent. claimed for these machines by Mr. Rice. If the results of that test are reliable the advantage of the reciprocating engine is enormously increased.

Referring to Mr. Rice's final computations, he adds together the total operating cost of a plant of turbo blowers and turbo generators as against a complete gas-driven plant, thereby securing for the turbo blowers a part of the admitted advantage of the turbo generator. It may well be pointed out that the desirability of using a turbo generator should be discussed on its own merits and is entirely independent of the question of using the turbo blower. There is the best of reasons to anticipate that a plant composed of turbo generators and reciprocating blowing engines would prove superior to a plant made up exclusively of either turbine or reciprocating engines.

In regard to the fluctuations of pressure which Mr. Rice has shown,

these seem to me to be of an order of size so small that it would be impossible to deduce their effect on the blast furnace. It is true that in many respects the blast furnace is a very delicate apparatus, but that its operation is effected to the extent which Mr. Rice claims by so small a fluctuation of pressure as he shows to exist, is scarcely within the bounds of probability. Such superfluous refinements are like tying a micrometer caliper to the end of a 50-ft. cotton tape. It is, therefore, impossible to admit the validity of Mr. Rice's claim of increased output, less coke, and less flue dust per ton of iron.

I would like to say that I believe that there is a field for the use of the turbo blower. I, myself, have some patents which I regard as valuable, covering its use for blast-furnace purposes along certain lines, and should be the last to say that there was no room for the turbo blower in the blast-furnace business, but it seems to me certain that its field is narrower than that assigned to it by Mr. Rice. Where economy is a matter of importance, the turbo blower should only be adopted at new plants for reasons independent of economy, such as its small size and the small foundation it requires and its freedom from liability to cause pulsations to the earth and consequent vibration to other structures when installed on soft ground.

On the other hand, the turbo blower can be used to enormous advantage in reinforcing existing plants; using a low-pressure turbine, driven by exhaust steam from the existing blowing engines and driving a single-stage turbo, which pre-compresses the air, and delivers it to the inlet of the air cylinders at 6 or 7 lb. pressure. This greatly increases the pressure and volume which the engines can blow, increases the steam economy, and reduces the stresses in the engines.

KARL NIBECKER, Youngstown, Ohio.—The data as given are most interesting and instructive, but several values mentioned do not appear to check with the results we have encountered.

While the actual operating conditions under which we are working may be somewhat different from those encountered by Mr. Rice, I do not feel that they are vitally at variance with some of the practice which he has probably encountered. In order that the information may be of service in general, we beg leave to call attention to some points at which we have found differences from the conditions as mentioned.

On p. 722, Mr. Rice speaks of the air delivered by the blowing engine as often being only 70 or 80 per cent. of the displacement of the piston. While this may be true in some cases, I do not feel that with a modern engine, in a reasonable state of repair, there is any justification for the volumetric efficiency being less than 90 per cent. The only conditions under which I have encountered the low volumetric efficiency mentioned by Mr. Rice is in very old engines or engines which have not been given proper attention.

On p. 727 he states that the superior steadiness of the blast from the turbo blower results in less loss in fuel and ore, being evidenced by the reduction in the amount of flue dust. While this may be the case under certain conditions, it must not be taken too literally. When a furnace is blown with a turbo blower it is probably easier to drive the furnace somewhat harder and the increase of production obtained will probably be attended with an increased production of flue dust and a consequent reduction in yield. Our experience with the turbo blower would tend to carry out this assumption. The amount of dust produced would also seem to be dependent upon the kind of ore used. From our practice with Mesabi ores, we are inclined to believe that more dust may be produced by the blower than by the engine under the same conditions of working and production.

On p. 735 Mr. Rice gives a value of 65 cu. ft. per minute as the engine displacement per pound of coke. The term cubic feet per minute is probably a typographical error, as the value should doubtless be given as 65 cu. ft. per pound coke. We believe that 53.5 would probably be a better value for modern practice. In case even as low a volumetric efficiency as 90 per cent. is assumed, 55 cu. ft. displacement per pound of coke should certainly be ample. We have repeatedly obtained indicator cards from an engine blowing a blast furnace producing an average production of better than 450 tons per day and found the indicated horsepower to be not in excess of 2,200 hp. In order to check the amount of displaced air per pound of coke with this horsepower, the calculation should appear as follows; we then have for the wind blown and horsepower developed, using 53.5 cu. ft. of air per pound of coke:

$$\frac{450 \times 2,240 \times 53.5}{24 \times 60} = 37,500 \text{ cu. ft. per minute}$$

$$\frac{5 \times 375}{0.94 \times 0.92} = 2,170 \text{ shaft horsepower.}$$

This checks well with tests made on blowing engines operating under conditions similar to the assumptions made in the paper.

The figure of 53.5 cu. ft. displacement per pound of coke, except in rare cases, would seem to be a very fair value and the above calculation checks with our practice.

In assuming 500° F. for the temperature of the gas delivered to the boilers, we feel that a very high value has been taken. Allowing 200° loss in radiation in piping and gas cleaning, this would require a furnace-top temperature of 700° F., and the gas must either be dry cleaned or not cleaned at all. If no cleaning is done, it would be practically impossible to maintain an efficiency of 70 per cent. in the boilers. We will, however, assume that the temperature of 700° F. is possible.

The gas required for boilers for reciprocating blowing engines would be as follows:

$$\frac{2,170 \times 14.15 \times 1,111.5}{103 \times 0.70} = 473,000 \text{ cu. ft. gas per hour.}$$

If the above value of 2,170 shaft horsepower is used for the gas engine, the gas per hour required would be:

$$\frac{2,545}{0.21} \times \frac{2,170}{95} = 277,000 \text{ cu. ft. gas per hour.}$$

The assumption of steam conditions, efficiency, etc., made in calculating the gas per hour required by the turbo blower is good power-plant practice. I believe, however, that most blast-furnace plants would hesitate to install equipment for this pressure and superheat.

If the blower is figured on 150 lb. steam pressure and 28 in. vacuum, which I believe is more in accordance with present blast-furnace practice, the calculation appears as follows:

Assume 12 lb. of steam per shaft horsepower.

$$\frac{5 \times 350}{0.70} = 2,500 \text{ shaft horsepower.}$$

The heat per pound of steam is determined as follows:

Initial total heat.....	1,194.6
Final heat of liquid.....	69.1
	1,125.5

$$\frac{2,500 \times 12 \times 1,125.5}{103 \times 0.70} = 468,000$$

Summarizing the gas requirements of the installations in cubic feet per hour, we have:

Reciprocating Steam Engine	Reciprocating Gas Engine	Turbo Blower
473,000	277,000	468,000

In the summary, the turbo blower shows a slight advantage over the reciprocating steam engine in gas consumption. This saving would be increased, due to the higher charges of investment, maintenance, depreciation, and attendance made against the engine. This difference in gas consumption may be very easily entirely eliminated by the excess dust produced when using Northern ores.

In conclusion, I would like to ask Mr. Rice how changes of atmospheric temperature and pressure affect the regulation of the machine; i.e., if the temperature or pressure of the air changes and the governor setting is the same, will the amount of free air delivered by the blower vary more than that delivered by the engine under similar conditions?

Considering the governing device as a restricted area having the usual thermodynamic formula of velocities,

$$V = C \left(\frac{p_1}{\gamma_1} \right)^{\frac{1}{2}} \left[\frac{2gk}{k-1} \frac{1 - \left(\frac{p_2}{p_1} \right)^{\frac{k-1}{k}}}{1 - \left(\frac{F_2}{F_1} \right)^2 \left(\frac{p_2}{p_1} \right)^{\frac{2}{k}}} \right]^{\frac{1}{2}}$$

and the ordinary formula for impact being

$$E = \frac{Mv^2}{2}$$

the impact by the disk becomes

$$E = CF_2 \left(\frac{p_1}{\gamma_1} \right) \left(\frac{p_2}{p_1} \right)^{\frac{1}{k}} \left(\frac{1}{p_1 \gamma_1} \right)^{\frac{1}{2}} \left[\frac{2gk}{k-1} \frac{1 - \left(\frac{p_2}{p_1} \right)^{\frac{k-1}{k}}}{1 - \left(\frac{F_2}{F_1} \right)^2 \left(\frac{p_2}{p_1} \right)^{\frac{2}{k}}} \right]^{\frac{1}{2}}$$

In which,

E = Energy

C = Constant

F₁ = Area in square feet of upstream section

F₂ = Area in square feet of disk

p₁ = Absolute pressure at upstream section

p₂ = Absolute pressure at disk

γ₁ = Weight of 1 cu. ft. of gas at upstream section

It then appears as though the effect of temperature and pressure changes may be exceedingly complicated and quite different from what might be expected from a superficial study of the governor.

W. TRINKS,* Pittsburgh, Pa.—Mr. Rice's paper on turbo blowers is doubtless interesting, but as I view it, mining engineers are particularly interested in machinery concerning mining operations directly, and for this reason I shall not only speak about turbo blowers, but also about turbo compressors.

On a trip which I recently made through England I found that one of the foremost engineering firms of Great Britain had built a number of turbo blowers for blast furnaces, but had built only a disappointingly small number of turbo compressors for mining purposes, whereas that same firm was a very extensive builder of reciprocating compressors for mines. Upon inquiry as to what caused the difference between these two products, and what made the turbo blower applicable in England when so few were used elsewhere, I was informed that the turbo blower can be used in England for blast furnaces on account of the following facts:

- (1) The ores are lumpy and open.

* Non-member.

- (2) Coal is very cheap.
- (3) The blast furnaces are low.
- (4) The rate of driving the blast furnace is very slow compared to American practice, so that the pressure is low.

On the other hand, with air compressors the pressure is high, fuel prices are high at most mines except at coal mines, and last, the demand for air is irregular. It is a well-known fact that turbo blowers, from a certain percentage of rated delivery downward, show heavy pulsations and surging, which make the operation irregular and uneconomical.

Originally I had intended to limit my discussion to the foregoing remarks, but statements made here by other parties compel me to add to what I said before. First, I wish to speak about the statement which was made concerning the cost of installation of the turbo blower being so much lower than that of the blowing engine that the bankers would compel engineers to give preference to the turbo blower. With this I disagree, because I am convinced that any comparison leading to the above given conclusion is made on the basis of engines as they were built 10 to 15 years ago. Engines built at that time run even now at a piston speed of 300 ft. per minute, whereas blowing engines built within the last five years are running at piston speeds of 600 to 800 ft. per minute. When it is remembered that the cost of a blowing engine per unit of air delivered is almost inversely proportional to the piston speed, it is evident that a modern blowing engine costs a great deal less than the old-time slow-speed blowing engine. This is information for the banker. Furthermore, blowing engines are more efficient now than they were formerly. Blowing engines with mechanically operated valves get out of adjustment, as may be proved by the fact that in one case with which I am familiar three such blowing engines which ran at 30 rev. per minute produced no more wind than one modern blowing engine of the same size running at 72 rev. per minute. Evidently the cost of wind production from the standpoint of first cost only had in this case been divided by three, although only reciprocating blowers were employed in the plant.

The statement was made that blowing equipment would have to pass through the same course of events as power-generating equipment; that is to say, turbo machinery would take the place of reciprocating machinery. I disagree with this statement also because there are two fundamental physical differences between the steam turbine and the turbo blower. The first one is that turbo machinery, such as a steam turbine, is most efficient at pressures below the atmosphere, and that the engine is more efficient at pressures above the atmosphere. Now, in the turbo blower the pressure range below the atmosphere is absent; that is to say, the blower must start to work at that pressure where it is less efficient than the reciprocating machine. The second reason for the lower efficiency of the turbo blower lies in the great difference be-

tween the conversion of pressure into velocity and the conversion of velocity into pressure. The steam turbine converts pressure into velocity, which is a highly efficient process. The turbo blower attempts to convert velocity into pressure and this is a very different proposition, as most of us know who have tried it.

The question of wear of the reciprocating engine compared to that of the turbo blower has been brought up. While I have not had personal experience with turbo blowers, I have had a great deal of experience with centrifugal pumps, and find that to throw the same quantity of water against the same pressure the speed must be gradually increased, because the leakage through the wearing rings increases with time. It is possible that the same leakage may occur in turbo blowers.

Finally, I wish to point out the great difference of opinion which exists between different engineers on the question of blowing equipment for furnaces. Whenever Mr. West or Mr. Freyn reads a paper, he proves conclusively that the gas engine is the only equipment that can be considered for blowing purposes and that furnace owners cannot afford to install a turbo blower, even if it were given to them. On the other hand Mr. Rice proves just as conclusively that the turbo blower is the only right equipment and that gas engines cannot be considered in any event. From this standpoint, we should welcome the suggestion, which I believe was made by Mr. Iversen, that an unbiased committee should be appointed to investigate the relative merits of the two types of blowing machines.

OTTO BANNER,* Phillipsburg, N. J.—There is practically no wear at all on the packing rings of turbo compressors or blowers; they stay unchanged just as the bearings of steam turbines do. They do not wear as they do in centrifugal pumps. In the case of one turbo compressor, these rings, after an eight years' run, were found to be fully as tight as when the compressor was started.

Turbo blowers in Germany are in use only on isolated and small blast-furnace plants, the tendency in that country being to prefer the gas-engine blower. If for a new machine steam has to be used, turbo blowers, and not reciprocating blowers, would be installed.

The fight between the gas and the steam engine is to a large extent decided by the cost of coal, but of late a remarkable development has taken place in Germany. They are installing in their newest plants gas-driven blowers, but steam-turbine driven electric generators. It would go too far to explain the reasons for this, but it is also very remarkable that in these new plants they install steam-turbine driven turbo blowers as reserves.

The turbo compressor for 85 to 100 lb. pressure has an undisputed

* Non-member.

field over there in the coal mines, first mixed-pressure and now live-steam turbines being chiefly used. The sizes of these units have been going up very rapidly during the last few years.

I advocate very strongly Mr. Iversen's proposition to nominate an independent commission to investigate exactly the merits of the reciprocating and the turbo blower. In Germany we have been up to this same controversy in the early days of the turbo compressor when it became necessary to find means for measuring the air input or output of these machines. I have been using low-pressure nozzles of varying sizes and the pressure difference used was usually not more than 20 in. of water column. The shape of these nozzles was that of a vena contracta; they have been adopted by the Society of German Engineers as standard for measuring actual delivered or taken-in air for reciprocating and turbo blowers and compressors, after a series of very exhaustive tests to find out the efficiency of these nozzles. For carrying out these tests, a big gasometer was used.

I wish these nozzles would be adopted in this country also because their use would be fair to builder and user, and comparisons of all these different engines would be possible on the same basis of actually delivered air. Their use is extremely simple, as is also the figuring out of the test results. The Ingersoll-Rand Co. has been using these nozzles during the past few years on its reciprocating and turbo compressors and blowers. We have found, for instance, that on a 100-lb. reciprocating compressor the actually discharged volume reduced to intake conditions was 89 per cent., the indicator card showing 93 per cent.

J. E. JOHNSON, JR.—My remarks were the result of an absolutely impartial discussion. I have no connection with any manufacturer of engines or turbo blowers; what I said was the result of an investigation that was impartial. I have been interested in the subject for a number of years.

H. S. BRAMAN, Youngstown, Ohio.—The papers have been somewhat disappointing, due to the fact that they have been dealing with the furnace man's side of the question. We carried on an experiment a short time back, which I would not want to go on record as calling absolutely correct, with two blast furnaces of exactly the same size, using the same percentage of Mesabi ores, and practically the same fuel, one equipped with a turbo blower, and the other with a reciprocating engine. The air necessary to make a ton of iron in one furnace was 4,362 lb. with the turbo blower; on the furnace with the reciprocating engine 4,062 lb. of air, a difference of 300 lb.

Another thing we noticed with the reciprocating engine is that the furnace makes dust only during times of high pressure, but on the turbo blower, the furnace makes dust all the time, high or low pressure.

J. E. JOHNSON, JR.—The figures I gave as to steam consumptions, were based on tests made by mining companies, and made on an engine built by one of the best engine builders of the country, and anybody who desires to get an engine from that builder, can get one with a guarantee that it will give better steam consumptions than I have quoted, and that guarantee will be backed up by the whole capital of that corporation. There is no question about the validity of those figures.

ARTHUR G. MCKEE, Cleveland, Ohio.—Since the installation of the first turbo blowers, I have been trying with the resources at my command to find out the facts in regard to the advantages of turbo blowers versus steam-driven reciprocating blowers and gas-driven reciprocating blowers, and I am frank to admit that up to the present time I have not been able to get anything which is conclusive and satisfactory in my mind in regard to this matter.

The discussion this morning has been somewhat partisan, which is bound to be the case in a large measure in discussions of this sort. However, there are some of us who are absolutely disinterested. We want to find out what is the best blower for the different conditions, regarding possible vacuums, fuel cost, and a number of other items which enter into this matter in a vital way, and I hope that something will be done by the members of the Institute with this in view, that we may get accurate non-partisan information about this whole matter. It is of great importance to the owners and builders of blast furnaces to know what is the best installation under each combination of conditions, taking into consideration the fuel cost, the vacuum which is available, the depreciation cost, the operating cost, and not overlooking the first cost, which is in some cases more or less lost sight of, but is of very great importance in providing blowing equipment.

RICHARD H. RICE (communication to the Secretary*).—The critics of the paper all base their remarks on two fundamental errors. The first of these consists in the ignoring altogether of *volumetric efficiency* as applied to a blast-furnace blowing engine, or in attempting to measure it from an indicator card of the blowing cylinder. The second is in considering all turbo blowers as alike in operative characteristics.

As to the first error, it arises in a confusion of definition. The term "volumetric efficiency" as used by me means the ratio of air actually discharged from the engine to the displacement swept through by the pistons, expressed in terms of cubic feet of free air at an inlet temperature of 60° F. P. 722 of the paper sets forth clearly the sources of loss of this efficiency and a slight study of these will be sufficient to show the im-

* Received Mar. 8, 1915.

possibility of measuring this efficiency by an indicator card. The only way is by actual test as therein stated.

On p. 732 of the paper the difficulties of measuring the discharge from a reciprocating blowing engine are set forth. Since the paper was presented F. G. Cutler has read a valuable paper dealing with tests made on reciprocating blowing tubs to obtain the volumetric efficiency (American Iron and Steel Institute, Birmingham meeting, October, 1914), in which he shows that, using his method of throttling the discharge to reduce the pulsations of the flow, the engines tested have a volumetric efficiency of about 80 per cent. I believe that Mr. Cutler did not fully succeed in eliminating the irregularities of the flow, and that if anything this efficiency is too high. Since then, tests that I have made comparing turbo blowers and the latest design of reciprocating blowing tubs indicate about the same figure which Mr. Cutler gives as the value of this efficiency. This being so, the figures of Mr. Iversen are quite incorrect.

Mr. Johnson's speculations as to what would happen if a certain type of reciprocating steam engine having a certain assumed steam consumption (which I believe is incorrectly stated) were put to drive a blowing engine, are interesting; but such an engine, by reason of complicated mechanism, small cylinder clearance spaces, etc., is not adapted for the rough service of blast-furnace power plants.

Mr. Johnson's statement that a test has been made at the turbo-blower plant of the Iroquois Iron Co. is, I am informed, incorrect.

If it be inadmissible to study the details of blast-furnace operation in order to determine ways of improving the practice, then my tests on fluctuation of blast pressure need not be considered. It is, however, by just such studies that advance in the art must be made.

Among those who fall into the second error which I mentioned are Messrs. Nibecker and Braman. Discussing the operation of an apparatus not fitted with an adequate air governor, they reach the conclusion that the turbo blower makes more dust and produces less iron. Both these effects are precisely those which would result from overblowing the furnace and I have no doubt that this is what has been done.

Reports from the furnaces blown by the apparatus described in the paper indicate an amount of dust produced rather less than normal and an increase in the production, effects which are exactly opposite to those reported by Messrs. Nibecker and Braman.

The Plant of the Duplex Process for Making Steel

Discussion of the paper of J. K. FURST, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2493 to 2514.

W. MCA. JOHNSON, New York, N. Y.—On discussing this paper I am open to the criticism of "bringing coals to Newcastle," as I, a zinc man, am bringing new ideas on steel to Pittsburgh. But still the view of an outsider can be valuable to the insider. Any multistage process, where each apparatus operates on its particular part of the work at a high efficiency, constitutes an advance in the state of art. But in multiplicity there can be complication and ensuing loss and inefficiency.

The duplex steel process is not the pronounced success which was predicted for it by its adherents, for several of such plants have reverted to plain open-hearth methods.

If we regard broadly refining processes, such as the Bessemer, open-hearth, puddling, copper refining, lead cupelling, nickel refining, we find that in general refining processes are intermittent and concentration processes are continuous. (This generalization I believe to be original with me; it was communicated to Dr. E. F. Roeber, editor, and appeared as an editorial in *Metallurgical and Chemical Engineering*, March, 1905.)

Where we desire primarily quality in a metallurgical operation we must put a batch of metal in the furnace, work on it until we get it to the required purity, test it, and take it out; whereas, if we desire primarily quantity we shove ore into a furnace and smelt it as fast as possible, allowing conditions to make a non-uniform product provided only we get tonnage.

From general commercial principles, I believe that the weak point of the duplex process is that the operation is not a money maker unless the plant is kept at 90 to 95 per cent. of its rated capacity, or unless the Bessemerizing is done so often that it is practically continuous. For in some manner the operation should be made continuous and the control on the tonnage should be commercially flexible. The commercial rigidity of having to control purity exactly increases operating costs without a corresponding gain. Conversely, the chief reason for failure of "steel from ore direct" is that it makes a final operation continuous, which should be naturally intermittent.

ARTHUR G. MCKEE, Cleveland, Ohio.—I would like to ask a couple of questions in regard to this paper, and in asking them I appreciate the difficulty that Mr. Furst has encountered in preparing a paper of this sort and in getting the information, which really, to my mind, is the

important information to the owner of a plant for the producing of steel, and also to the man who is responsible for the building of such a plant.

In the first sentence is stated:

"The reasons for manufacturing steel by the duplex process are, briefly: saving of time; increasing output for capital invested; and avoiding the difficulty sometimes experienced in obtaining scrap."

That is briefly stated, surely, but how effective is such a plant in giving the results referred to in that statement?

At the close of the paper, Mr. Furst remarks:

"The benefits derived from the duplex process for making steel have been enumerated in the reasons given for duplexing; but we may say in conclusion that the process has brought about the development of the Bessemer converting plant to a higher state of perfection and efficiency, has created a desire for larger and better hot-metal cars and mixers, and has aided those practicing it to solve some of their problems. Since the process has been practiced in this country for seven years only and is still in an imperfect state, it is hoped that some of the master minds working on it will succeed in bringing about a more perfect stage of development within the near succeeding years, and thus another step will have been taken toward the uplift of mankind."

Does the process make, as claimed, a larger tonnage per day, and approximately how much?

Does it reduce the cost per ton?

Does it increase the production per dollar of plant investment, and if so, how much?

And, also, another very pertinent question: How does this process affect the quality of the product, if at all?

Mr. Furst doubtless does not have an opportunity to get these facts, but can somebody else enlighten the Institute, so that we can know from actual experience in operation what conditions justify the installation of the equipment required for the use of this process; what profit and other advantages will be obtained; in short, whether it has justified the claims made for it.

HENRY D. HIBBARD, Plainfield, N. J.—Regarding Mr. Johnson's remarks, when you come to make a finished product, such as steel, you want to take a batch of material and keep it in hand until it is in proper shape and then cast it. Where you are running for a crude product, as in the case of the blast furnace, it is all right to run continuously, but when you want a finished product, to try to avoid taking a batch and keeping it under control until it is done, will be a mistake. In the open-hearth furnace we take a charge and keep it until it is done; in the Bessemer, we have, to some extent, to catch it on the fly; but in each of these cases the charge is treated until it is as near as we can get it to what we want when it is cast. Ever since the Bessemer process was established, there have been proposals to make steel by treating iron with blast as it moved forward, and some of the pyrotechnic displays occasioned thereby have

been very wonderful. Such plans are all failures. Within a few years patents have been taken out for processes of that description, but I think no progress has been made with any of them.

As to the output of the duplex process, I think it varies from about three open-hearth heats of finished steel to about 20 heats in 24 hr. There are furnaces in this country that have produced over 1,000 tons of steel a day and have kept it up for a month. On the other hand, three open-hearth heats of 50 tons each would give an output of 150 tons a day.

BRADLEY STOUGHTON, New York, N. Y.—As to the question whether the duplex process increases the output, it is possible to answer this in either of two ways:

The output of the duplex process, when properly operated, is much more than the output of the open-hearth process alone, but is not so great as the combined output of the open-hearth process and the Bessemer process each working independently of the other. This, however, is not the point really involved, because the output of the two processes working independently has now no longer the same industrial possibilities, because we have not in this country sufficient Bessemer ore to operate the Bessemer process economically and up to its capacity. But when the two processes work together they can use ores now available in this country in very great quantities, which are above the Bessemer limit, and can produce a good quality of steel.

As to the question whether the interest on the investment required to install the duplex process is more than compensated for by the increased profits, we can answer this by inference through our knowledge that several companies which are wisely managed, and which years ago abandoned the Bessemer process, have recently re-installed converters in order to employ them in the duplex process.

I believe that the duplex process has very decided commercial and industrial advantages, and hope that those who are operating it will co-operate with this Institute by coming forward and exchanging their information with others in order that benefit may be accorded to all.

The Iron Industry in Brazil

Discussion of the paper of E. C. HARDER, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2573 to 2586.

I. C. WHITE, Morgantown, W. Va.—I have seen some of the great iron-ore deposits described in this paper. One of these in the State of Minas Geraes appears to be an immense mountain of iron ore about 2,000 ft. in height. All over its surface from bottom to top may be seen large masses of rich iron ore, varying in weight from a few pounds to several tons, and from all that one can judge without exploitation, the quantity of available ore is certainly very large in this particular mountain. Whether it will prove a mere surface deposit or shell, like the famous Iron Mountain of Missouri, is for future exploration to determine.

The question of transportation is the main factor in determining the availability for exportation of these Brazilian iron ores. There is no coal in Brazil that can be manufactured into coke without expensive previous treatment, since the best of the raw coal in the States of Rio Grande do Sul, Santa Catharina, and Parana, contains about 35 per cent. of ash, of which about 6 per cent. is sulphur, and coal of this composition cannot be utilized in iron smelting without purification.

When I was chief of the Brazilian Coal Commission in 1904 to 1906, I had a cargo of this coal shipped to Kalk, Germany, and treated in the great coal-testing plant of the Humboldt Engineering Works, located just across the Rhine from Cologne. It was found that by crushing and washing the Brazilian coal, it yielded 33 per cent. of briquetting coal, containing only 10 to 12 per cent. of ash (of which 1 per cent. was sulphur), and 42 per cent. of slack coal, with 25 per cent. of ash and low in sulphur, so that about 33 per cent. of the Brazilian coal could be converted into coke of fair quality, as shown by the following average of three analyses, using air-dried coal, of briquets made from the Barro Branco coal bed, State of Santa Catharina:

Moisture.....	1.56
Volatile matter.....	31.66
Fixed carbon.....	56.73
Ash.....	10.05
Total.....	100.00
Sulphur.....	1.34
Phosphorus.....	0.003
B.t.u.....	13,395

A briquet from the Sao Jeronymo coal, State of Rio Grande do Sul, gave the following results on analysis:

Moisture.....	5.03
Volatile matter.....	33.42
Fixed carbon.....	50.77
Ash.....	10.78
Total.....	100.00
Sulphur.....	0.61
Phosphorus.....	0.033
B.t.u.....	12,496

The briquets were, of course, made from crushed and washed Brazilian coal to which about 5 per cent. of pitch had been added as a binder. These coals when crushed and washed would yield a fairly good coke, but nevertheless the all-important question of transportation would remain to be solved, since the coal deposits are about 1,000 miles distant from the ore and limestone deposits and railway transportation in Brazil from the coast to the interior and *vice versa* is very expensive. The main reason for this is that a great wall of granite 2,000 to 3,000 ft. high parallels the coast from Bahia southward to Florianapolis or beyond, through which the rivers descend in cataracts, and thus have not yet graded out easily accessible railway routes from the seacoast to the interior, so that railways are compelled to climb up over this granite wall with steep grades, driving tunnels and making deep cuts and rock shelves along high cliffs at vast expense, the funds for which have been provided mostly by foreign capital at high interest rates. The scenery on the journey from Paranagua up to Curityba in the State of Parana cannot be excelled for rugged grandeur and wildness on any of the railways of Switzerland. The great forests of Parana, into which this railway enters, cannot compete, however, with lumber from Florida and Russia in the markets of the coastal cities of Brazil, although transported for thousands of miles instead of hundreds, owing to the transportation costs from the mountain region to the seacoast, a distance of less than 100 miles. Hence, whether it will be possible in the near future to bring the Brazilian ores 400 to 500 miles from the interior to the seaboard at a cost low enough to make their exportation to the United States or to Europe feasible is a question that has not yet been solved; and it certainly will not be in this generation, when good coal brings about \$10 a ton by the shipload delivered at the seaport cities of Brazil, and \$15 to \$20 per ton when delivered at the ore mines of the interior of the country.

At one locality in the State of Minas Geraes, there occurs such an assemblage of minerals necessary in the manufacture of iron and steel as probably occurs nowhere else in the world; namely, a deposit of excellent limestone standing nearly vertical, upon one side of which is a deposit of good iron ore, many feet in thickness, and on the other side 20 ft. of good manganese, which is mined and shipped to the United States. It might be possible to ship coke from the United States to this

rich mineral zone of Brazil, and thus inaugurate a Brazilian iron and steel industry, since at the time of my travels there (1904 to 1906) only one small charcoal furnace making 5 to 6 tons a week was in existence in the entire area of that great republic.

E. C. HARDER (communication to the Secretary*).—Replying to Dr. I. C. White's discussion. While some of the Brazilian iron-ore deposits are of great size it is only rarely that an entire mountain or hill is composed of ore. As the iron ores occur in beds and these beds are generally of considerable hardness they frequently form ridges, the shapes of which depend upon the attitude of the layers. Where the beds are gently inclined the dip slope of a ridge is usually gradual, while the opposite slope is steep and frequently precipitous, due to more rapid erosion of softer rocks underlying the capping bed of iron ore. In such a case the iron-ore bed usually extends down the dip slope for varying distances, depending upon its horizontal extent. It will also extend some distance down the precipitous slope, depending on the thickness of the bed, while below the iron ore the underlying rocks appear, such as itabirite or ferruginous schist. In many places these underlying rocks are concealed by a thick talus of rubble ore broken and fallen from the upper slope. The fragments forming the talus vary in size up to many tons.

In places where the bedding is vertical or nearly so hard iron-ore layers may form a sharp, steep-sided ridge, of which the ore forms the crest while the slopes are formed by softer adjacent rocks. In this case also the slopes are generally covered by a mantle of ore fragments.

The Brazilian deposits differ from those of Iron Mountain, Missouri, in that they are sedimentary beds or lenses like limestone or shale, while those of Iron Mountain are veins in igneous rocks. Naturally the Brazilian ores may be expected to extend to a greater depth down the dip, since sedimentary beds are in general much more continuous than veins.

With regard to transportation, the question is one of distance rather than high rates. The government railway (Central of Brazil Ry.) has taken contracts to transport iron ore from the ore district to Rio de Janeiro, a distance of 300 to 400 miles, for 6s. (about \$1.50) per ton, not a high rate. The principal difficulty lies in the fact that most of the ore will have to be hauled this great distance by rail to reach the coast and then, after trans-shipment, will be subjected to about 5,000 miles of ocean transportation to reach North America or Europe. Thus only the very high-grade, hard ores would seem to warrant the expense of transportation at the present time.

As stated in the original paper, it seems improbable that the

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Central of Brazil Ry. on account of sharp curves and prohibitive grades, not taking rates into consideration, will be able to transport more than a small percentage of the ore which must eventually be taken from the district. For this reason another railroad, the Victoria to Minas, is being constructed, which, instead of going south to Rio de Janeiro, as the Central of Brazil Ry. does, goes in an easterly direction and reaches the ocean near Victoria, a small port 300 miles northeast of Rio de Janeiro. This railroad will follow the Rio Doce on a down grade for the entire distance to the ocean and no part of it will have a steeper grade than 1 per cent. Over a railroad thus constructed it should be possible to transport ore with a minimum of expense.

Dr. White suggests that it should be possible to bring coal to the iron-ore fields and smelt ore at a profit there, since both limestone and manganese ore are available right in the iron-ore district. While it is true that manganese ore occurs locally in the district and that in some places impure limestone occurs, these materials are after all of minor importance and the great necessary item is coal or coke. To ship coal to South America for the purpose of smelting iron ore and then returning the iron products to Europe seems wasteful and quite unwarranted. Even if some portion of the product were utilized in South America, why should it be more advantageous to bring the coal to the iron-ore fields than to take the iron ore to the coal fields and manufacturing centers? In the history of iron mining the natural trend has been for the ore to go to the coal fields and for manufacturing centers to be established there. This we see in case of the Spanish and Swedish ores which go to England and Germany for smelting. This also has been the history of the Lake Superior ores, which have gone, and in large part are still going, to the Pennsylvania coal region for smelting. It is only within recent years that coal is being taken West for iron-ore smelting purposes and that extensive smelting operations have been started at Gary and West Duluth. It has been found more profitable to transport iron ore to established manufacturing centers and to return the finished products to the original source than to manufacture these articles at the source. Thus in Brazil at the present time the limited number of articles of iron manufactured with cheap labor and from raw materials, such as wood and iron ore, which cost but little, find it difficult to compete with similar articles imported from Europe and North America.

The Brazilian iron-ore problem has been carefully studied during the last few years by men competent to judge, and the prevailing conclusion has been that the ore can be transported to Europe to be smelted and the products sold at a profit, whereas it would be quite out of the question to attempt the establishment of an extensive iron industry in Brazil.

The Oil Fields of Mexico

Discussion of the paper of EZEQUIEL ORDOÑEZ, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2531 to 2536.

DAVID T. DAY, Washington, D. C.—Dr. Ordoñez is really the pioneer in the study of the occurrence of Mexican oil. His studies came at such an early time as to prove really prophetic. He was successful in pointing out the relationship of these oil deposits to the volcanic teeth which are so very characteristic as a geological feature in the coastal plain of Mexico. At that time only the first wells around Ebano had been drilled, and drilled under the direction of Dr. Ordoñez himself. Only these Ebano wells were available for a real study of the condition of the occurrence of oil, the other large gushers not then being known. From that time on, the study of the geology of the country has, in large measure, been able to keep up with the oil development. As usual, a great deal of haphazard work in the exploration of the oil fields has been carried on. The results have been good, bad, and indifferent. We know a great deal more about the distribution of oil in Mexico than we do about the geology of it. Two ideas have become prevalent in regard to the Mexican oil, due to the peculiar development there: (1) That oil is omnipresent over a tremendous area bounded by De Soto La Marina, in the province of Tamaulipas, on the north, to south of Tuxpam in Veracruz; and (2) that there are very few spots in all this region which have rich gushers. The truth is somewhere between the two. The amount of land taken up, chiefly by three or four large companies, is extremely great, and covers all the possible chances. The idea that oil is going to be discovered in a large percentage of the land taken up by the oil companies, seems to me entirely unjustified. The amount of oil that will be produced from comparatively small spots here and there over that territory and other territory, undoubtedly will be very great. I do not think there is any hope of multiplying the capacity of these large gushers by the area of that part of Mexico in which they are contained, as a large number of promoters are inclined to do.

Another view is that these tremendous gushers will last a short while, then each well will be drowned out by salt water. Many people have really condemned a large and profitable field on account of that. Considering these areas from the north southward, the first large area is the Panuco region, where the Corona well of the Dutch syndicate is found. Nobody has much idea of what that well yielded before it became convenient to measure it. The large pond of oil which this Corona well yielded was measured with a great deal of froth on it. The oil is very

heavy, contains some water and froths a great deal. There is very much doubt that the product went up to 200,000 bbl. a day, as some claimed. Its yield was probably somewhere between 30,000 and 50,000 bbl. a day. This Panuco region has been very prolific in large gushers and gives every evidence that an area of at least 10 square miles will be quite productive. At least 2,000 acres in that region can be considered as proved oil territory of great value. No trouble has been experienced with water, but the oil is very thick. It is a peculiar class of oil, and will be valuable when a large fuel-oil trade is developed and facilities for handling it are afforded. But there is more oil now than there are facilities or market for it.

East of Panuco there is another interesting region, the Topila field, about half way between Tampico and the Panuco field. There, also, some good gushers have been discovered, but because of the influx of water have been prematurely condemned. That is unfortunate, because nearly every one of those gushers was very profitable before it went to the water. Yet an acre of that region is valued at perhaps one-tenth or one-twentieth part as much as in the Panuco region. Considering the price, I believe more oil will be produced for the money invested in the Topila region than in the Panuco region.

Farther south are the large gushers which have shown not only enormous daily capacity but also staying qualities, and are unlike any other wells in the world of very large size. Those are the wells at Casiano and Potrero del Llano.

As regards the geology, a few of these regions have been developed by geological study; the rest have been found more or less by accident. In the Panuco region there are only slight evidences of anticlinal structure, by which a rational development of the oil territory might be planned. When it comes to Topila, after all is said, nothing is known about the geology of it or the relation of the water stratum to that of the oil below. When that is known, very probably the oil can be successfully taken out without running into the water. It will require much more geological study than anybody has yet put on it.

It is interesting to know that in the Potrero del Llano region the great No. 4 well of the Aguila Company was really discovered by careful geological study, which showed a structural dome in a topographic basin. A number of small wells yielded the geological information and enabled Dr. Hayes to develop the big well strictly from geological data.

PHILIP W. HENRY, New York, N. Y.—Mr. Ordoñez's paper brings to mind my experience in that part of Mexico 10 years ago—prior to its development as an oil field—in making an investigation of the residue from oil seepages, such residue being known locally as *chapapote* and in the paving trade as "natural asphalt." Up to that time the only natural asphalts which had been developed commercially were the two

deposits, situated 100 miles apart, known as the Pitch Lake in the island of Trinidad, B. W. I., and the Bermudez Lake in Venezuela. Both of these "lakes" are composed of the residue of asphaltic petroleums, the lighter oils having been distilled by the sun and wind. In the Pitch Lake of Trinidad the petroleum has filled the crater of an extinct mud volcano having a surface area of 110 acres, and in the Bermudez Lake it has spread over the surface of a swamp to the extent of 1,100 acres. From borings which I made in the Pitch Lake in 1894 the depth is at least 150 ft. in the center, which was the greatest depth attainable with the boring apparatus at hand. A boring made in the lake several hundred feet from the edge showed a bituminous sand at a depth of 90 ft., indicating a bowl-shaped crater. The surplus seepage or asphalt has run down the sides of the crater—the lowest edge of which is 110 ft. above sea level—into the Gulf of Paria about $\frac{3}{4}$ mile distant. Owing to the large amount of asphalt taken from the lake during the past 25 years, from 100,000 to 200,000 tons per year, the present level of the lake, which settles uniformly, but slowly, as a whole, is now some 10 or 15 ft. below the lowest edge of the crater. Seepage is still in progress, but probably at a rate of not more than 5,000 tons per year. The depth of the Bermudez Lake is about 10 ft., and seepage there also still continues. At the present rate of consumption both of these lakes or deposits are practically inexhaustible. In the Pitch Lake of Trinidad the petroleum has become so intimately mixed with water and earth that the residue or crude asphalt has a remarkably uniform composition of 40 per cent. bitumen, 28 per cent. water, and 32 per cent. fine sand and clay, finer even than Portland cement. In the Bermudez Lake the residue or crude asphalt has taken up a certain amount of surface water and earth, varying from 5 to 40 per cent. of water and 2 to 5 per cent. of earth. As refining consists merely in driving off the water, Trinidad refined asphalt contains 55 per cent. of bitumen and Bermudez asphalt about 95 per cent.

These two deposits were originally developed by separate business interests, which made a combination in 1894, thus forming a monopoly in the supply of natural asphalt. This caused a search for other natural asphalt deposits, or oil seepages, among which were those in Mexico lying between the Panuco and Tuxpam rivers, 120 miles apart. This district during the past four years has developed into the greatest oil field of the world. In 1898 to 1900 asphalt was taken from a deposit known as Chapapote Núñez, some 2 miles north of the Tuxpam River, and from Cerro Viejo, 8 miles north, and used in paving certain streets in Chicago and Jersey City. This asphalt was dug from the deposits, placed in boxes holding about 250 lb. each, floated 60 miles down the Tuxpam River and then loaded on steamers lying outside the bar. The amount thus shipped was small, probably not over 2,000 tons in all, and the expense was prohibitive, both on account of the small amount of

asphalt in the deposit and the cost of handling and transportation. In 1902, A. H. Carner, who had originally developed the Bermudez asphalt deposit, but who at that time was no longer connected with it, became interested with A. L. Barber in these deposits or seepages in Mexico, and obtained oil and asphalt leases to the extent of 80,000 acres scattered between San Geronimo on the Tamiahua Lagoon, 60 miles south of Tampico, and Cerro Azul, 30 miles to the southwest. In order to rid the crude asphalt (petroleum residue) of water and earth, he erected a small refinery at San Geronimo, within half a mile of the famous (later) Des Bocas well, and produced a refined asphalt running about 95 per cent. of bitumen, quite similar to Bermudez. The seepages near San Geronimo being soon exhausted, he brought *chapapote* or crude asphalt from the Juan Casiano lease, 10 miles distant. In all, it is not likely that more than 1,000 tons of asphalt were shipped from San Geronimo, most of it being used in the City of Mexico. In the fall of 1904 I was engaged to make a careful determination of the amount of asphalt on these leases and the cost of building a railway to connect the largest deposits with the Tamiahua Lagoon. A cursory examination soon showed that the only deposits of any magnitude were those at Cerro Azul, which could be connected with the lagoon by building a railroad 20 miles in length to Cuchillos on the Tancochin River, and a survey of this route was made.

At Cerro Azul, the crude petroleum, instead of coming up through a mud volcano as at Trinidad or spreading over a swamp as at Bermudez, ran down water courses, and the seepage was so small that, although still going on, there had accumulated during the centuries not more than a depth of 3 ft. of residue or asphalt in any one place. In fact, the seepage was so much less than at Trinidad or Bermudez that the total accumulation in this district was trifling in comparison. To determine the exact amount of asphalt, I took cross-sections every 50 ft., making soundings every 10 or 20 ft. on the section by means of an axe or machete, thus determining the amount in each deposit. Within a radius of a mile I found 14 separate deposits worthy of examination, and the following table shows the average length, breadth, and depth of each deposit, together with the area, volume, and tonnage of asphalt, figuring 70 lb. per cubic foot.

As this amount of crude asphalt would not produce more than a total of 20,000 tons of refined asphalt, it was evident that as compared with the Trinidad and Bermudez deposits, which were practically inexhaustible, supplying 150,000 and 30,000 tons respectively per year to the trade, these deposits in Mexico were negligible as far as the business of the world was concerned.

For use in Mexico, however, with its heavy duty on foreign asphalts, there was a market, which in connection with the development of oil

Deposit	Length, Feet	Average Breadth, Feet	Average Depth, Feet	Area, Square Feet	Volume, Cubic Feet	Tons
A	1,900	87.6	1.07	166,525	177,999	6,230
B	400	172.9	0.44	69,175	30,506	1,068
C	800	32.6	0.73	26,100	19,186	672
D	350	107.9	0.28	37,750	10,451	366
E	200	64.5	0.32	12,900	4,106	144
F	1,470	13.8	0.38	20,300	7,600	266
G	1,050	47.1	0.44	49,450	22,030	771
H	1,800	158.1	0.76	284,650	216,006	7,560
I	600	121.2	0.56	72,730	40,116	1,404
J	1,450	93.1	0.94	135,025	126,855	4,440
K	350	75.1	0.72	26,275	18,946	663
L	500	79.8	0.53	39,400	20,962	734
M	1,200	31.6	0.75	37,880	28,868	1,011
N	440	95.5	0.65	42,010	27,117	949
Total....	12,510	81.5	0.73	1,020,170	750,768	26,278

and of agriculture would have warranted the construction of the railroad. Just then, however (1905), Mr. Barber and Mr. Carner again took control of the Bermudez deposit, and as their interest was in asphalt rather than in oil, in 1906 they sold their leases in Mexico to E. L. Doheny and associates, and these properties are now among the most valuable belonging to the Huasteca Petroleum Co. A narrow-gauge railroad 30 miles in length now connects these properties, extending from Cerro Azul to San Geronimo. On the Juan Casiano lease is the famous well, which during the past four years has produced some 34,000,000 bbl. of oil, and is still producing at the rate of 23,000 bbl. per day. Nothing, however, has been done with these deposits of natural asphalt, for as Mexican crude petroleum, like California crude, has an asphaltic base, there can be manufactured out of it an excellent paving asphalt, which is now being used in such important cities as New York, Philadelphia, and Baltimore. During the present year asphalt manufactured from Mexican crude petroleum has been used in paving Fifth Avenue, New York City, between 26th and 34th Streets, thus entering into serious competition with Trinidad and Bermudez natural asphalts. In the development of the oil fields of Mexico, more particularly the district between the Panuco and Tuxpam Rivers, it is interesting to remember that it was the oil seepage (natural asphalt) rather than the oil itself which attracted certain capital to this region eight years before its remarkable value as an oil field had been demonstrated.

I. C. WHITE, Morgantown, W. Va.—On my first visit to the Mexican oil fields, in 1911, I was accompanied by Señor Ordoñez, who was the first one to predict the presence of great oil deposits in Mexico and whose

intimate acquaintance with Mexican geology assisted me greatly in obtaining a clear understanding of the vast oil fields of our sister republic. It may not be generally known that the most productive oil well ever yet drilled in the world's history is located in Mexico, and known as Casiano No. 7, situated on the coastal plain about 75 miles south of Tampico, 25 miles west from the Gulf, and probably 20 miles from the famous Dos Bocas gusher. The boring is at the foot of a volcanic hill, 250 to 300 ft. in height, and near where a large seepage or petroleum spring had been issuing through cracks or crevices in the rocks made by the welling up of these geologically recent necks and bosses of diabasic lavas, thus creating the dome-like structure in the underlying Tamosopa limestones, the mother rock of the oil, so favorable to the accumulation of oil and gas deposits in commercial quantity. This well, which is owned by the Huasteca Petroleum Co., a subsidiary of the Mexican Petroleum Co., was struck on Sept. 11, 1910, and up to Sept. 11, 1914, when it was just 4 years old, has produced, according to a telegram just received from E. L. Doheny, President of the company, 33,580,000 bbl., and is still yielding, at the rate of 20,000 bbl. daily, oil of 22° gravity Baumé without any trace of water, while at the same time emitting daily about 10,000,000 to 12,000,000 cu. ft. of natural gas. When one considers the space necessary to hold such an enormous mass of liquid (196,250,000 cu. ft.) and 100,000,000,000 cu. ft. of natural gas, even when under the original rock pressure of 600 lb. to the square inch, it is realized that the total space required to hold this amount of oil and gas would total nearly a cubic mile, and that hence the conditions of the rock reservoir or oil pools in the Mexican fields must be completely dissimilar to those anywhere in North America, or anywhere else in the world for that matter. In all other oil fields of the world, so far as known, the deposits of petroleum and natural gas are stored principally in the pores and minute spaces of coarse sand and gravel or crystalline dolomite without the intervention of any large cavities, but in the case of these Mexican wells that have produced such enormous quantities of oil, as Casiano No. 7, Potrero del Llano No. 4 and others, it is unbelievable that vast open spaces do not exist in the underground limestone reservoirs of these great Mexican gushers. Of course the upward bulging of diabasic necks, bosses, and sills of lava would open large fissures in the limestone, and the accompanying acidulated waters would doubtless form additional solution cavities, the evidence of which Dr. C. W. Hayes, of the Mexican Eagle Co., thinks he has seen in specimens of limestone thrown out of his company's great Potrero del Llano well No. 4, located about 30 miles west of Casiano No. 7. This famous well of the Mexican Eagle Co. is probably the largest oil well ever discovered, since reliable estimates of its production when first struck place the output at 150,000 to 160,000 bbl. daily, all of which passed down the Buena Vista and Tuxpam rivers into the Gulf of Mexico

for a period of 54 days, at the end of which time the flow was turned into an earthen reservoir 20 to 30 ft. in depth and covering an area of 65 acres. It then by actual gauge registered a production of 100,000 bbl. daily. This great well was finally gotten under control, and successfully shut in, although the rock pressure was 850 lb. to the square inch. This was done in 1911, the well having been drilled unexpectedly into the oil horizon about Christmas, 1910. For the last two or three years, it has been permitted to produce 25,000 to 30,000 bbl. daily, or as much as the pipe lines could transport to the loading stations on Lake Tamiahua, and also to the Gulf of Mexico near Tuxpam, where the oil is loaded by a combination of pipe line and rubber hose directly into ocean steamers in the open Gulf. Reports from Mexico state that this famous well was entirely closed in during the recent revolutionary troubles in the Tuxpam region, and that within the last few weeks the oil escaping in large quantity from seepages through fissures in the vicinity of the well, had been set on fire, possibly by lightning. The well is therefore in great danger from the intense heat, which has already destroyed the packing around the joints of pipe lines leading directly from the well, thus permitting a vast amount of leakage of oil, which was also burning. The well itself is securely shut in under a large mound of reinforced concrete and when I saw it in April, 1913, there were no indications of extensive seepages of oil anywhere near the well.

When Casiano No. 7 was struck, it was shut in entirely after it had flowed for a few hours at the rate of 50,000 bbl. daily. When the oil began to spout out of the ground over three or four acres around the well at the rate of 4,000 or 5,000 bbl. daily, the General Manager, Mr. Wiley, opened the valve which closed the well in, until the rock pressure was reduced to only 290 lb. to the square inch instead of 600, the rock pressure of the well when entirely closed in. The oil then ceased to spout through the ground, but went through the pipe line into a wet-weather pond on the borders of Lake Tamiahua until adequate tankage could be rushed to completion. It is probable that the pressure in the Potrero del Llano well could not thus be relieved on account of lack of storage reservoirs during the revolutionary troubles, although at present, in spite of the great fire raging near the well from the leaking lead lines and seepages, other lead lines not affected by the fire are reported to be delivering 30,000 to 35,000 bbl. of oil into tankage to be pumped away to tidewater.

With such wells as these two monsters, and the area of possible oil territory in Mexico only as yet scratched, as it were, it is almost certain that in productive capacity of low-grade (22° gravity Baumé) fuel oil, our sister republic is destined to surpass California and all other known fuel-oil regions of the world.

Refining Oil by Liquefied Sulphur Dioxide

Discussion of the paper of DR. L. EDELEANU, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2313 to 2332.

F. W. BUSHONG,* Pittsburgh, Pa.—This process of extraction is particularly advantageous in the case of Roumanian oil, on account of its high aromatic content. I have prepared a paper, which will appear probably next month in the *Journal of Industrial and Engineering Chemistry*, describing a series of two-degree fractions of oil from the Nowata field in Oklahoma. I have made extractions with liquid sulphur dioxide in accordance with Dr. Edeleanu's method, and found that the Oklahoma oils, for instance the kerosene fractions, are purified in this way, although the quantity taken out does not at all compare with that taken out of the Roumanian oil. I have made careful analyses of separate fractions, before and after extraction, and they show a very distinct fractional separation. I like the distinction brought out by Dr. Day in previous discussions, in regard to oils, rather than oil, and his beautiful illustration of fractionation. Just as fuller's earth, and other clays, will enable us to obtain lower-boiling fractional portions of the oils, and eventually retain a residuum which is hard to get rid of, or to remove, so there is an advantage to be gained by the use of a reagent which begins at the other end by taking out these last constituents first, namely, ozone. I have also analyzed the ozonides and they are derivatives of hydrocarbons very much poorer in hydrogen than we have heretofore supposed. Similar results have been obtained with Roumanian, Russian, and Italian petroleum. Furthermore, I have prepared ozonides from a fraction of Oklahoma oils, both before and after extraction with liquid sulphur dioxide, and found that the ozonide obtained from the extract is the same as that precipitated out of the original oils. In other words, the residual substance, which the fuller's earth retains, is the one first thrown down by the use of ozone, but it may also be extracted by liquid sulphur dioxide and be thus separated in apparently unaltered condition. The analysis of this extract corresponds to the formula $C_{15}H_{20}$, which is surprising, since it is lower in hydrogen content than we have been in the habit of considering among the constituents of our oils. The extract was probably contaminated with other hydrocarbons, but I think it is unquestionable that we must look for hydrocarbons in petroleum which correspond to a formula at least as low in hydrogen as $C_{15}H_{20}$, and possibly even as low as $C_{15}H_{16}$.

* Non-member.

Chlorides in Oil-field Waters

Reply to discussion of the paper of C. W. Washburne, presented at the New York meeting, February, 1914 (*Trans.*, xlviii, 687 to 694 (1914)).

C. W. WASHBURNE, New York, N. Y. (communication to the Secretary*).—Professor Lane makes an interesting contribution to the study of chloride waters, in saying that calcium chloride waters occur not only in the greenstones of the Lake Superior copper mines, but also in the interbedded felsitic conglomerates and "in far-removed sandstones, *e.g.*, near Whitefish lake, and Ontonagon, in the Michigan iron mines, and in the Storm King granite." Surely these are not connate waters preserved from the Algonkian sea, assuming that the Algonkian sediments are marine rather than continental. Finding the same type of water in the Storm King granite points rather toward an abyssal source. The possible connection of these waters with the magma of the greenstone is immaterial, although some basic calcic magma is the most probable source. It is to be expected that the ascending water of the region would be found not only in the greenstones, but in all the rocks which it could penetrate.

The ascent of water of this kind along vertical passages, and the consequent addition of a common ion to the circulating sedimentary salt solutions, would explain the precipitation and localization of the "giant, concretionary" salt plugs of the Gulf Coast, without making the excessive demand for an improbable quantity of salt in the adjacent sediments that is required by the unmodified theory of Harris. Without such a precipitating medium most of the dissolved sodium chloride would have been lost by passing on to the surface without precipitation. The base to which the chlorine ions were attached is immaterial, providing they were sufficiently concentrated, but the presence of the chlorides of calcium and magnesium in the solutions is indicated by the associated bodies of secondary dolomite.

Volcanic free chlorine or hydrochloric acid could not migrate far without picking up bases, which would be largely magnesium and calcium

* Received Mar. 6, 1915.

¹ J. C. Branner and R. N. Brackett: *American Journal of Science*, 3d ser., vol. xxxviii, No. 223, p. 50 (July, 1889).

G. F. Kuns and H. S. Washington: *Trans.*, xxxix, 169 to 176 (1908).

A. H. Purdue: *Economic Geology*, vol. iii, No. 6, p. 525 (Aug.-Sept., 1908).

Include references to diamonds.

if the source were in underlying plugs of magnesio-calcic magmas. Rocks of this type reach the surface near by in the peridotite dikes and pipes cutting Cretaceous strata at the inner margin of the coastal belt in southeastern Arkansas,¹ and in the many dikes and plugs of olivine basalt cutting Cretaceous and Eocene strata in Mexico, along several hundred miles of the coastal belt itself. Where the stratigraphic cover is thicker, and the fracturing apparently less intense, as in Louisiana, coastal Texas, and southern Mexico, it is quite reasonable to believe that the intrusions failed to reach the surface and are deeply buried, manifesting themselves only in the peculiar phenomena of the salt plugs of these terminal regions.

Besides the Arkansas and Mexican intrusions, there are peridotite dikes of unknown age, Triassic or later, cutting Carboniferous strata in Elliott and Crittendon counties, western Kentucky,² one of which extends into southern Illinois. Also there are dikes of similar peridotite at Syracuse³ and at Ithaca,⁴ New York. It is suggestive that all of the intrusions in or near the eastern group of American oil fields are of this ultra-basic type. Wagner⁵ says that the New York, Kentucky, and Arkansas peridotites are essentially identical with the diamond-bearing kimberlites of South Africa. Sir Henry Roscoe⁶ extracted aromatic hydrocarbons from the kimberlite of the Kimberley mine. In the Kentucky peridotite, or kimberlite, as Wagner calls it, Diller recently has found particles of metallic iron suggesting the carbide-bearing iron in a basic phase of similar rock at Ovivak, Greenland. The Arkansas peridotites carry diamonds. All of the Mexican intrusions that I examined, some 20 or more, contained oil. In Cuba there is a striking association of oil with serpentine, including many occurrences. The appeal to magmas of peridotite nature is therefore not far fetched. In many respects these rocks resemble the meteorites in which hydrocarbons have been found, which are supposed to be fragments of disrupted asteroids or other heavenly bodies. The inorganic nature of the meteoric hydrocarbons is, of course, unquestionable.

From these and other arguments it seems that the hypothesis of

¹ J. S. Diller: *American Journal of Science*, 3d ser., vol. xxxii, No. 188, pp. 121 to 125 (Aug., 1886); *Bulletin* No. 38, *U. S. Geological Survey* (1887); *American Journal of Science*, 3d ser., vol. xiv, No. 262, p. 286 (Oct., 1892).

² G. H. Williams: *American Journal of Science*, 3d ser., vol. xxxiv, No. 200, p. 137 (Aug., 1887); *Bulletin of the Geological Society of America*, vol. 1, p. 533 (1889); *Science* (Mar. 11, 1897).

C. H. Smyth, Jr.: *American Journal of Science*, 4th ser., vol. xiv, No. 79, p. 26 (July, 1902).

⁴ George C. Matson: *Journal of Geology*, vol. xiii, No. 3, p. 264 (Apr.-May, 1905).

⁵ P. A. Wagner: *The Diamond Fields of Southern Africa*, p. 106 (Johannesburg, 1914).

⁶ *Proceedings of the Manchester Literary and Philosophical Society*, vol. xxiv, p. 5 (1885).

the igneous origin of oil has sufficient elements of probability to be worthy of consideration, at least tentatively. The method of multiple working hypotheses, developed by Chamberlin,⁷ is the only scientific way to attack a problem of this kind, in order to be *able to perceive* all of its various significant features. In this way we have the best opportunity of finding conclusive scientific evidence of the origin of oil, which is completely lacking to-day. It is possible, moreover, that the conflicting evidence will ultimately be harmonized by the conclusion that some crude oils are of organic, others of inorganic, origin. The conservative student will reserve judgment until a complete theory is developed and tested in the field and laboratory.

Every one will agree with Hunt, Becker, Lane and others, that the composition of the ocean must have been changed by contributions from the rivers, also that juvenile and volcanic waters may have affected it, especially in pre-Cambrian time. Under the planetesimal hypothesis of Chamberlin,⁸ all surface waters were derived originally from the interior of the earth, and the composition of the primitive ocean would be nearly that of the juvenile waters of the time. The calcium chloride waters in the Algonkian and other rocks of the Lake Superior region conceivably may be of this nature, if they have risen from the Archean. Whether they are called "connate-Archean" or "juvenile" is immaterial.

We do not know the original composition of the ocean and there are only hypotheses of the rate and character of the changes it has undergone. However, under any hypothesis, there has not been sufficient time since the Paleozoic to account for the amount of change in sea salts that is required by the theory that, in the deep wells under consideration, the waters are connate with the strata now containing them. Considering only the ratio of Na:Cl, some of the well waters would represent an ocean only one-fourth, others one-half, as old as the present sea. If allowance be made for any sodium in the original sea water they would be correspondingly older. Under the connate-water hypothesis, they should occur, therefore, in the pre-Cambrian rather than in late Paleozoic formations.

The composition of the Paleozoic sea water is largely hypothetical. It has been suggested, on biological grounds, that it must have exerted approximately the same osmotic pressure as to-day in order to have supported the same types of life. If so, the pressure was maintained by greater dissociation, because the Paleozoic sea water must have been more dilute than now. Yet the very deep well waters are concentrated brines, one of them having about seven times the salinity of modern sea water. To apply the connate hypothesis to these waters, it is necessary to assume that they were concentrated by evaporation,

⁷ *Journal of Geology*, vol. v, No. 8, pp. 837 to 848 (Nov.-Dec., 1897).

⁸ Chamberlin and Salisbury: *Geology*, vol. ii. See also R. T. Chamberlin, *The Gases in Rocks*, *Carnegie Institution Publication No. 106*, pp. 70 to 75 (1908).

either before or after burial in the sediments. The former would be possible locally, where shallow, semi-detached embayments and small inclosed seas permitted great evaporation, but it could hardly explain the frequency of deep concentrated waters in many places. The latter method, concentration in the rocks, is possible on the assumption that there are rising rock gases, which drive the water out and evaporate that left behind in the deeper rock pores. Neither of these hypotheses has much strength at present, and I have hesitated to suggest the latter, which if carried far would choke the passages with mineral matter. This would explain the cessation of the ascent of water in ore veins, etc., but it is doubtful that the choking of the passages would not occur too early to permit the effective operation of the process. More likely there is also water rising with the gases and keeping the passages open, but in any case it seems likely that these deep ascending waters are strong solutions. The dryness of the deep rocks can be explained possibly better by the process of capillary concentration, described in another paper.⁹ After the cessation of water circulation, by the exhaustion of the magmatic or other source, or the closing of communication, the nearly stagnant water may have been transferred to the finest pores and joints by capillary exchange with the rock gases, which would be concentrated in the larger openings. The water would not flow readily from these minute capillaries into mine workings.

The theory of non-migratory connate waters fails especially to account for the wide variation in the ratio of Na : Cl, as well as its great defect below any ratio possible in late Paleozoic times. A modified theory would be more acceptable, by assuming that the deep well waters have risen from different horizons of originally connate waters in the basal Paleozoic and pre-Cambrian rocks, in which the ratio of Na : Cl might be as low as in the well waters. Thus they would not be connate with the strata now containing them, which they would have invaded from below.

The evidence of the analyses consists in an excess of chlorine with reference to sodium which is far above anything possible in stagnant connate waters. The simplest explanation is that there has been a direct contribution of calcium and magnesium chlorides, the salts to which the present excess chlorine must be assigned in calculating the analyses. This would be effected by the ascent across the strata of water resembling that in the ancient rocks of the Lake Superior region, and its mixture with overlying connate waters.

In examining the analyses of Ohio waters I was struck by the number of analyses in which the ratio of Na : Cl is almost the same as in the water of the existing ocean. In some analyses not only this ratio but also the

⁹ The Capillary Concentration of Gas and Oil, *Bulletin* No. 93, Sept., 1914, pp. 2365 to 2378.

percentages of sodium and of chlorine in the total salts are almost the same as in modern sea water. These must be true connate waters and they indicate a surprisingly small subsequent modification in sea water. In examining the few analyses of waters in oil fields in Tertiary strata I failed to find any which did not appear to be greatly contaminated with the local circulating ground water. It seemed unfair and meaningless to use the brines from the salt plugs of the Gulf Coast, although they present some interesting features.

Professor Lane doubtless has studied this question much more thoroughly, and I await with interest his paper on the chemical evolution of the ocean. It will be surprising, however, if he can manage to explain the great and widely varying excess of chlorine in the water of the deep wells of Pennsylvania and Ohio, by the unmodified connate hypothesis. His seas must have changed remarkably in composition from place to place, unless he admits with me that the waters may have risen from different horizons of connate water in deeper strata, or else that there must have been extensive contributions of more deep-seated (juvenile?) waters in calcium and magnesium chlorides. In either case an ascent across the strata is required.

Professor Lane finds "a serious difficulty" in getting the chloride waters across the St. Peter sandstone, because where that formation can be drilled it now contains a comparatively fresh artesian water. However, it should be noted that wells in the oil fields cannot reach the St. Peter sandstone, and that we do not know the character of the water in that formation beneath the oil pools. From our experience in the deep wells of oil fields we may suspect that the water of the St. Peter sandstone beneath the oil fields may not have the same character as in the areas of artesian circulation, but that it contains either a brine or a sulphur water, or more probably that it is dry at such great depths.

An identical case is furnished by the Dakota sandstone of Colorado, which commonly contains an artesian circulation of somewhat alkaline but comparatively fresh sulphato-carbonate water similar to the common ground water of the outcrop region along the front of the Rocky Mountains. Yet the same sandstone contains an entirely different water beneath the Florence oil field; namely, a hot water, highly charged with hydrogen sulphide and with salts, in which I believe the sulphates and chlorides¹⁰ are most conspicuous. This water was encountered in a well drilled near the southeast margin of the field, where the strong solution quickly corroded away the iron casing, and also in the well on the northwest side of the field which now supplies the Canyon City swimming pool. Similar hot sulphur water escapes from springs in the margin of

¹⁰ Written in the field, from memory, without notes: subject to error in the character of the salts.

the granite west of Canyon City, pointing to its source. If we had only the evidence of the artesian wells in the Dakota sandstone of the surrounding region, we would never suspect the existence of this type of water in the same formation beneath the Florence oil field. It is quite possible to make the same mistake in connection with the St. Peter sandstone.

The localized occurrence of the hot sulphur water in the Dakota sandstone of the Florence oil field is probably not without significance, because sulphur waters are generally characteristic of the Rocky Mountain oil fields, especially in Wyoming, and of many other oil-bearing regions; *e.g.*, the Apscheron peninsula, the Gulf of Mexico, the deep sands of California, etc. Sulphur has an association with oil similar to that of chlorine, which at once suggests the fact, discovered by students of ore deposits, that sulphur and chlorine are the principal mineralizers of the basic magmas, as distinguished from fluorine, etc., for the acid magmas. It is to ultra-basic magmas that the supporters of the volcanic hypothesis must look for the source of oil. The true reason for the association may be far different from the one suggested, but that it is a real association, not without some significance, cannot be doubted by any one familiar with the phenomena of the different oil fields of the world, in all of which, so far as I know, one or both of these substances, sulphur and chlorine, are conspicuous.

I would like to ask the students of ore deposits how they explain the fact that the ground water appears to have a lower limit. The joints cannot be tightly closed at the depths attained by mines. Why has the water ceased to flow at these depths where it formerly deposited minerals? Why should it be so concentrated at some deep places, as in the lower levels of Lake Superior copper mines? Do you think the water could have been shoved out by ascending rock gases? Have you any pressure records, or analyses, of deep-mine gases that suggest an outward creep, or diffusion, of abyssal gas, such as that furnished by the helium of the Kansas natural gas? Can you apply the process of capillary concentration to explain the dryness of the deep fissures? It seems to me that you have some problems in common with those who are studying oil fields, and that it would be profitable for us to get together with our various lines of evidence and our different methods.

The Capillary Concentration of Gas and Oil

Discussion of the paper of C. W. WASHBURN, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2365 to 2378.

ROSSELL H. JOHNSON, Pittsburgh, Pa.—It seems to me rather important to bear in mind one thing which is sometimes lost sight of, that the marine sedimentary rocks are laid down in water. We have to start with an initial charge of water. That being so, and the surface tension between the water and the oil being as slight as it is, as referred to by Mr. Washburne, the phenomena of capillarity are bound to be most important as between the liquid surface and the gas surface. So far as the liquid-gas surface is concerned, capillarity, in my opinion, is of the utmost importance, and it seems to me this paper of Mr. Washburne's is of great value to us for that reason. Starting out as we have, with our attention concentrated upon gravitational sorting, we have had to wait for the discussion of movement furnished by Munn, and this discussion of capillarity by Washburne, for a more balanced view of the situation. It seems to me that in the accumulation of oil and gas we have a problem of the interworking of gravitational sorting, capillarity, viscosity, immiscibility, and movement. All five things seem to me to be important. I would have expected Mr. Washburne to have given a fuller treatment of the relation of movement to capillarity. In my own experiments I find that gravitation and capillarity alone seem remarkably impotent when everything is at rest, but if I furnish a little movement, gravitation and capillarity step in and accomplish a great many results that one would not expect. Take a horizontal tube and fill a portion at the middle with an ordinary sand saturated with oil, and then put water and sand in each end, and the tube may remain at rest a very long time without that oil distributing itself along the top of the tube. But with a little motion that oil will gradually work up to the upper side. So gravitation and capillarity alone are rather impotent unless they get this element of movement. It would not be necessary to have very extensive movement, but some degree of movement is essential.

Now if natural gas, in addition to the method of origin by the action of bacteria on organic matter in mud, may also arise by pressure at great depth, as the overburden is added, a great deal of motion may result. I think more motion is produced in this way than by the descent of meteoric waters.

The author, I think very properly, calls attention to the fact that acceptance of the rôle of capillarity still leaves us using the anticline and the structure contour in prospecting. It seems to me very important

that gas men should realize that their relation to the anticlinal guidance is on a different basis from that of the oil man. If anticlinal prospecting has a certain degree of validity for oil, it has a far higher degree of validity for gas, so that the absolute necessity for gas men taking into consideration the anticlinal structure seems to me very important.

When I wrote the article referred to by Mr. Washburne, I was skeptical for the reasons given as to whether the difference between oil-gas and water-gas surfaces in surface tension could be of much importance. Yet, having observed in the laboratory how a drop of gas in water becomes so quickly surrounded by oil if there is any oil at hand, and strongly believing in the great importance of capillarity as between fluid and gas, a method by which capillarity would indirectly affect oil was suggested.

There is another practical consideration in prospecting that depends upon this theoretical question. Does oil arise in sand or in shale? These views indicate very strongly in my mind that there is no longer that necessity for its arising in the sand that we have had pressed upon us by some authors, particularly English. If the oil has to be indigenous in sand, the best section for our prospecting would be one with a maximum of it. We should have a large percentage of sand contributing to our ideal section. On the contrary, our best oil fields always have a great preponderance of shale, with merely interpolated beds of sand.

D. T. DAY, Washington, D. C.—I think it is perfectly clear that the study of Mr. Washburne, concerning the accumulation of oil by capillarity in the earth, views the matter somewhat as a wick connecting that oil with something else: Oil will flow through and accumulate at the other end of the wick. But such an accumulation would not develop any particular pressure. Mr. Washburne has hardly laid stress enough on the very great pressure which may result where the oil has been driven out of a shale by the superior capillary force of water. In that way a very great pressure can be obtained.

Mr. Washburne's paper, following a great deal of practical work that has been done by a number of persons, including myself, is a very long theoretical step in advance, which will serve the particular purpose of clearing the atmosphere and developing a theory which will be easy for others to think of and to apply practically. It will stimulate a great deal of practical experimentation, I hope.

I object to the way in which he has used the expression "friction" in his paper in several places. This idea of friction seems to be more exactly adhesion, than friction, which we must think of as always a function of motion. It is a function varying with the speed with which particles are moved; when the speed is zero, friction is zero. The average geologist forgets that he speaks of friction where the motion is infinitely slow and friction cannot exist.

This paper also should receive a modification, I think, in your minds in this way. The geologist is not necessarily a chemist, and the geologist will consider oil as oil. There is no such thing as oil. You must think of *oils*. And the very first thing that happens in the migration of oil through the earth is practically always a change in the character of the oil itself. That offers complications to this theoretical consideration. It also simplifies the matter, inasmuch as by the careful chemical study of these oils one can derive a great deal of information as to where the oil has been, and its relation to the other oils nearby. The study of earth movements of oils is going to be very much simplified by determining the differences in oils adjacent to each other. For example, in about 1896 it was my privilege to determine for the first time, I believe, that mineral oils can be changed in composition by diffusion through rocks, especially shales.

By taking a Pennsylvania oil, and carefully filtering it until it was a perfectly clear liquid, but still crude oil, I absorbed it in dry clay, practically such a plastic clay as fuller's earth. Shale from the region of Titusville, Pa., was also effective. The clay was then shaken up with water. Water drove out most of the oil but not all. It did not drive out the oil that went in, but it drove out different oils. Is that perfectly clear? You drive out some oil if you put in an amount of water equal to the volume of clay. If you put in more water, you will drive out more oil, but when you are through, oil will be left in the clay that cannot be driven out by the water. I do not say that in geological time we might not drive out every particle of that oil, but I do not believe so, and we have no proof that we can. If, instead of clay, you take finely powdered glass and absorb the oil with that, and treat that with water, you get nearly all the oil out in the same condition in which it went in. Two features are essential to the discussion of this matter: Mr. Washburne's article considers oil and space, no matter what the wall of the space is; the chemical nature of that wall is not considered. Neither is one oil considered as different from any other oil. Oil is oil. Those two things are fatal for a complete discussion of the theory of this subject, because one oil will work in one way, and another oil will work entirely in another way. In the first place, the kind of surface of the capillary tube exerts an important influence other than simply on the rate of capillary movement. That is shown best in this way: Suppose you allow an oil to be soaked up by a clay. Now add water and examine the oil that comes out, and examine the oil that is left in the clay. There are means by which you can get it out, but not by simple addition of water to it. Take those two oils. What are they? If you determine what proportion of the oil that went in was made up of unsaturated hydrocarbons, and determine the proportion of unsaturated hydrocarbons in the oil driven out by the water, you will find that the oil driven out by means of water has a much

less percentage of unsaturated hydrocarbons in it. If you go further and take the clay, from which all the oil that is possible has been driven by water, and try to dissolve out the oil with gasoline, some more of the oil can be dissolved out, but not all of it. By adding ether, some of that oil which is perfectly soluble in ether can be taken out. But ether will not take out all the oil which was perfectly soluble in ether before it was absorbed in the clay. Go one step further and try benzol. That will drive out some more, but not all of it. And as far as I have gone, I do not know of anything in the way of a solvent that will extract all of that oil out of the clay, although the oil was perfectly soluble before.

That just indicates that very possibly some of those shales which are supposed not to have any oil in them, because it cannot be extracted by the ordinary solvents, but only by destructive distillation, may still have oil there as such, in the same condition as the oil which has gone into a clay and cannot be brought out by means of any ordinary solvent. In other words, from the evidence of some 2,000 experiments carried on in this study by myself, it is perfectly evident that so soon as an oil containing mixtures of saturated and unsaturated hydrocarbons goes into a clay (not powdered glass, but clay) the capillary surfaces of clay act differently in respect to different oils. As soon as an oil goes into clay *fractionation takes place*. The unsaturated hydrocarbons cling to the surface of these capillaries and will not go on. The saturated hydrocarbons, interdissolved at first with the unsaturated hydrocarbons, go farther and pull away from the unsaturated material in solution with them. That holding back of a material by a surface has received a particular term that is not at all explanatory. It is simply naming that property as adsorption—not absorption. That word “adsorption” is used by Mr. Washburne here apparently meaning absorption, the two being confused by himself, or possibly by the printer. By adsorption these things can hang back. We have a familiar example of that in the manufacture of lake dyes. If an aniline dye is dissolved in water and alum added, a clear solution of the aniline dye and alum in water results. That dye will remain dissolved indefinitely. But as soon as ammonia is added, aluminum hydroxide is formed, which takes hold of the dye and leaves an absolutely clear water solution again, the dye entirely disappearing. This is an analogue of the property of clay surfaces of taking one oil out of solution in another, and holding it back.

This must be considered in connection with problems in the earth. We must remember that there are certain things in the way of oils that cannot go through shale at all. An asphalt oil from Texas will not pass through a shale until that shale has been entirely saturated with asphalt. Then the other light, unsaturated hydrocarbons will simply hang back until the clay surfaces are saturated by them and then finally they go

through and are not affected by these surfaces, except in respect of capillarity, as with saturated hydrocarbons.

We hope to simplify our problems very greatly by the study of the capillary concentration of oils. We must give careful consideration to the study of all these capillary phenomena, especially to the adsorptive effects on certain hydrocarbons, which undoubtedly have played a great rôle in this matter.

Turning to Mr. Johnson's discussion: he has enunciated a very important idea in regard to movement. It seems to me that when we look at the relations of oil and gas and the great deal of help effected by movement, we recognize the very considerable difference in solubility of natural gas in various oils. The old idea of a layer of water and a layer of oil and a layer of gas in contact with each other must be greatly modified, because under the pressures existing most of that natural gas must be in solution in the oil. A bottle of Vichy looks clear, but when the bottle is opened the gas escapes. It was there in solution in the water. So we have a larger reservoir of gas for a given pressure if the oil is present to dissolve that gas. In the case of oil with gas in solution, just as soon as the pressure on the oil is released the oil remains but the gas escapes. Any change of pressure on that oil must affect the gas in solution in the oil, so the cause for the movement of oil, in migration, is, it seems to me, perfectly self-contained. Any slight change of pressure or temperature will be sufficient to give plenty of reason for movement.

EDWIN M. CHANCE, Wilkes-Barre, Pa.—I am very glad Dr. Day touched upon the subject of adsorption. That, to my mind, is the most important consideration of this subject. Mr. Washburne in his paper has considered the action of the capillaries as purely mechanical. I am afraid that if we consider it from this angle only, we will see but a very small part of the picture. The nature of the capillary surfaces, as Dr. Day has well said, is of great importance, as the phenomena of adsorption will exercise, and, I believe, have exercised a great and far-reaching influence upon the nature, chemical composition, and general characteristics of the crude oils. The phenomena of adsorption are of every-day occurrence and very wide usefulness in the arts. The clarifying action of bone black, fuller's earth, and like substances, depends entirely upon this action. The clarification of drinking water by alum precipitation depends largely on this, and it is only reasonable to believe that when an oil penetrates gradually through great areas of rock, taking great time, and where the rocks are of a shaly nature, closely allied to what are known to be the best substances for adsorption phenomena, this trans-fusion must greatly affect the nature of the oil.

H. A. WHEELER, St. Louis, Mo.—The complications which Dr. Day's experiments certainly show are extremely important and are complica-

tions that vary with every clay; for I think Dr. Day will indorse my statement that you never find two clays alike in this respect.

To revert to this proposition solely as a mechanical factor, Mr. Washburne's paper would make the hydrostatic pressure entirely a factor of porosity. I would like to ask the members present if in a new field, before there has been any loss of pressure by drainage, the rock pressures have ever been found equal to or greater than the hydrostatic head due to the depth of the sand? In my own experience, I cannot recall a single instance where the pressure was not more or less under that which would be expected from the mere hydrostatic head; sometimes quite a marked shrinkage, which, with all due respect to Dr. Day, I will call friction loss.

G. A. BURRELL,* Pittsburgh, Pa.—Some years ago I made a few experiments at the request of Dr. Day, in which I dissolved natural gas in crude oils from California, to saturation. I found that at ordinary pressures and temperatures the oils absorbed from 15 to 25 per cent. of their volume of the natural gas of Pittsburgh. Casing head gases were absorbed to a greater extent, as much as 60 per cent. Under heavy pressures, of course, the solubility effect would be much greater.

I have been interested in the subject of the movement of oil and gas through the strata, principally in the movement of natural gas. The two are much related. We found that natural gases vary much in composition, as many others have noticed. We have, for instance, the so-called dry gas, or marsh gas, usually not found in localities where oil is present, at least usually not in as tremendous volumes as the gases that are present in the oil fields. In the oil fields we usually have a gas containing a higher percentage of the heavier paraffin hydrocarbons than methane. We were able to separate those hydrocarbons in the case of Pittsburgh natural gas and found that it contains 84.7 per cent. of methane, 9.4 per cent. of ethane, 3.0 per cent. of propane, and 1.3 per cent. of the higher paraffin hydrocarbons, chiefly butane. That is typical of the great quantity of gas used in Pittsburgh, Columbus, Cleveland, Cincinnati, and other places that obtain gas principally from the fields of West Virginia, though a great deal comes from Pennsylvania, and some from Ohio.

Passing from that type of gas to a gas in contact with oil in the stratum, we have found that the higher members of the paraffin series, higher than methane, predominate at times, and some of them contain enough of the vapors of the liquid paraffins to make them profitable as a source of gasoline.

Referring now to the movement of gas through sands. We have taken samples at different places in certain localities from different sands—

* Non-member.

that is, from a lower sand and an upper sand—and from the few data we have collected, we have found that the gas in the upper sand contains less of the heavier paraffin hydrocarbons than that in the lower sands, looking as if in moving from a point of common origin and passing up through strata and finally landing in the top strata, the gas had lost some of the more easily removable paraffin hydrocarbons.

We ran across an interesting case in the natural gas from the Hog-shooter pool, Oklahoma. That is a natural gas, containing about 95½ per cent. of methane, about 3½ per cent. of carbon dioxide, and the rest nitrogen. An unusual condition surrounding the gas lies in the fact that there is a tremendous body of gas containing methane as the paraffin hydrocarbon not far from oil sands where other paraffins than methanes are found.

If these gases came from a place of common origin with the other natural hydrocarbons, here is a case where in moving through a comparatively short distance, a tremendous quantity of gas had lost its higher paraffin hydrocarbons.

I wish to lay emphasis on the relation between the composition of natural gas and the presence of oil. As we find natural gas in the oil fields, it almost invariably contains the higher paraffin hydrocarbons, and the inference is that natural gas issuing from earth containing methane as the only paraffin hydrocarbon, may be usually taken as an indication that oil is not present in the immediate vicinity. In rare cases this may not follow.

D. T. DAY, Washington, D. C.—Our experimental data may be of great value perhaps in one connection. It has not been enough to go far on the chemical side of this question. It is principally on the geological side. We recognize that where oils contain unsaturated hydrocarbons they are, as a class, taken out by adsorption.

But suppose you only have paraffin hydrocarbons, what happens when they go through this clay? In the ordinary course of diffusion, the more viscous saturated hydrocarbons hang behind, so that you do get fractionation of paraffin oils. It is perfectly easy to show this by experiment. Taking a tube 6 ft. long, pack it tight with fuller's earth, put one end, open, into a receptacle containing oils from which all unsaturated compounds have been very carefully removed, and allow that oil to diffuse slowly through the clay and you will get gasoline at the top and vaseline at the bottom. You get it by a process that is purely diffusion. In this case there is no action of adsorption, so far as we know. But you do get fractionation by diffusion.

Professor Gurowicz, in commenting on the work I did in 1897, states that while I did effect those results I did not know what I was doing, and did not know why I got the results. He seems to think that there is no other action than adsorption. In the above case there was no separation

of any class of hydrocarbons; there was purely diffusion, having reference to the viscosity of the different oils. He does not consider the diffusion idea at all. He thinks it is entirely adsorption. I am prepared to combat that view. What Mr. Burrell has said here is sufficient evidence that that is going too far. Here, with a mixture of very volatile hydrocarbons, such as butane, propane, ethane, and methane, we evidently have separation by diffusion of those perfectly saturated hydrocarbons, so that eventually you have nothing but methane. It seems to be a perfectly clear case of separation by difference in viscosity. Just exactly as the original separation of hydrogen from oxygen by heat was made by the superior diffusibility of hydrogen. If you can fractionate two gases in that way, so you can fractionate two oils. This fractionation which Mr. Burrell speaks of seems to me to be exactly of the same class.

F. W. BUSHONG,* Pittsburgh, Pa.—When one takes an ordinary steel cylinder and fills it with a natural gas in the field, at high pressure, one finds on first opening it a very large excess of methane. The mixture follows the natural law of gas passing out through a small orifice. In that cylinder there will remain finally the high-boiling constituents. I wonder if Dr. Burrell has not observed that same thing in the passage of gas through the porous strata of the different fields, and I wonder whether this is not also a factor in the matter of hydrostatic conditions. The gas, of course, under high pressure will be dissolved in much larger quantities. But the point I want to ask him is whether that same field examined later on, after the enormous pressure has been relieved through consumption, will not show eventually a large amount of casing head gasoline and, like the Ohio and Indiana fields, will not yield almost entirely gasoline, where it now yields almost entirely pure methane.

G. A. BURRELL.—In the Hogshooter field the pressure is down pretty low; starting at 500 or 600 lb., it is now down to about 30 lb. I would think that if there was going to be a change in composition, it would have been noticed by now.

F. W. BUSHONG.—Have you observed such a change?

G. A. BURRELL.—Yes, that change takes place. The lighter paraffin gases have passed off in large measure in many oil wells.

C. W. WASHBURNE, New York, N. Y. (communication to the Secretary).—This reply to the discussion of my paper at the Pittsburgh meeting is based on the stenographer's transcript received in Africa.

I thank Dr. Waldo for his offer of new data on the surface tension of crude oils, which I hope he will add to this discussion at an early date. His observation that crude oil has surprisingly low surface tension, below

* Non-member.

† Received Mar. 6, 1915.

what would be expected from its specific gravity, is in accord with my determination of the surface tension of a Pennsylvania crude oil, mentioned in the paper. It accords also with my deductions from capillary theory.

I agree with Mr. Johnson in everything he says. The necessity for other movement, independent of capillary action, in order to enable the latter to effect concentration, is recognized in the paper, but the greater emphasis he gives the matter is helpful to correct understanding.

Mr. Johnson is correct also in objecting to the opinion of Höfer and other European geologists who believe that oil must originate in the same sands in which it is found and that it cannot penetrate shale. Clay is much better than sand for the preservation of organic matter and the great bodies of shale and limestone furnish a more adequate organic source of the oil which saturates the sands. Certainly in fissure fields, such as the Florence field, Colorado, and probably in some of the smaller Wyoming fields, in which there are no sands, it is absolutely necessary to admit that *the oil has passed through shale* in order to reach the short discontinuous fissures, or gashes, in the great shale formations. If the fissures at Florence reached the underlying Dakota sandstone, the field would have been flooded with water when the pressure was exhausted. It remains remarkably dry.

With Dr. Day I must differ. As shown in the paper, it is highly improbable that capillarity can cause any effective pressure whatever in the sands. The pressure in the closed tube, from which Dr. Day had driven the oil with water, may have been due to capillarity, but the mechanical conditions of the closed tube are not imitated in nature, except in fine passages that open into coarse ones, and which are closed at the other end. Capillarity would thus create pressure in the closed pores of shale opening into sand, but not in the sand.

Probably it is necessary to increase my estimate (15 atmospheres) of the maximum pressure which can be developed by capillary action. In the paper I called attention to the fact that "the minimum diameter of capillary openings is somewhat uncertain. It is usually placed at 0.0001 mm. for tubes and at 0.0002 mm. for plane fissures." These are the values given in Van Hise's *Treatise on Metamorphism*, in text-books, etc., based on the work of Lord Rayleigh and others, and are the basis of my calculation of maximum pressure. However, I have received a letter from John Johnston, of the Geophysical Laboratory, Washington, D. C., calling attention to the more recent work of G. Bakker¹; who considers that the radii of the spheres of molecular attraction are only six to seven times the molecular diameter. Consequently the length of the radii is only a few millimicrons, and capillary action can take place in pores very

¹ *Zeitschrift für physikalische Chemie*, vol. lxxx, No. 2, p. 129 (June 25, 1912).

much smaller than 0.0002 mm., developing pressures far in excess of those stated. This entirely justifies Johnston and Adams² in calculating capillary pressures for tubes as small as 0.01 micron, and invalidates my objection thereto. I have not had opportunity to examine the articles by Bakker (the only literature here being a missionary's bible!), but in this study it is not necessary to pay further attention to the thickness of capillary films and to the corresponding possible pressures in minimum capillary tubes, because the capillary pressure would be effective only in closed passages in shale and could not contribute to the pressure in the sand. Only a negligible pressure could be developed in the comparatively coarse pores of sandstone by the capillary withdrawal of water from fissures, and this would be more than overcome by the negative pressure left in the sand pores by the more general capillary pull of the water into shale. The capillary pressure of the basal water against the oil in the sand pores also would be negligible. There is no volumetric change involved.

Water would be drawn into the minimum capillaries of shale with great pressure, amounting to many hundred, possibly to a thousand, atmospheres, judging from Bakker's calculation of the sphere of molecular influence mentioned above. In rock pores the actual pressure would be of course very much less than that in the theoretical minimum capillaries. The pressure would be effective in driving the oil out of the minute pores of shale into joints and fissures and into the comparatively coarse pores of sandstone, but the attempt to explain pressure in the sand by such means is like the problem of a man lifting himself by his boot straps. Likewise, the attempt to explain the general migration of oil by capillarity or any continuous capillary movement, or any ascent of a liquid above the capillary height for the given pore diameter, is the problem of perpetual motion. These fallacies must be abandoned by students of oil.

My use of the term "friction of contact" was rather careless. Dr. Day is right in saying that friction is not merely adhesion, and that it increases with velocity, but he fails to note that in the little experiment in question it was necessary to consider only one element of friction: namely, the adhesion of drops of oil to the sides of an open vessel. This may be regarded as the initial element of friction, since it is the resistance to the first movement of a drop. This element is independent of velocity, and depends upon the area of solid contact, which in the pores of ordinary sands would be from 50 to 200 times the area of contact of the drops against the sides of the vessel. All other elements of friction, both internal (viscosity) and external, which come into play as soon as a drop begins to move, are to be added to the initial adhesion, and they strengthen the argument that gravity cannot lift drops of oil through a fine water-saturated sand. Gravity possibly may lift larger masses of light oil, as in

²*Journal of Geology*, vol. xxii, No. 1, p. 13 (Jan.-Feb., 1914).

Johnson's experiment, mentioned in this discussion, after circulatory or other movements of the water body have dislodged them, but not little drops, nor heavy viscous oil. Where the latter has been gathered in low anticlines, as in the Kern River field, the oil probably has increased afterward in gravity and in viscosity. The circulatory movements of the water would dislodge all of the drops in time, and these circulatory movements are much more efficient than gravity in carrying the drops and other oil bodies to the top of the water, where they would be held by the tension of the water-gas surface, whenever they happened to touch it. In very fine sands the process would be less efficient because of the small radius of curvature of the water-gas surface in the little pores.

The time available for the concentration of oil into sands may be very long, but it is not infinite. The time available for producing the anticlinal and other types of distribution of oil commonly is very short. The relation of some oil bodies to the ground surface and to the ground water indicates that the present distribution of the oil has been effected since the erosion of the existing topography. "Motion infinitely slow," therefore, would be ineffective, as it is in most geological processes, and hence in this study we cannot agree with Dr. Day when he says that here "friction does not exist" and that "it has reached the minimum limit." We cannot neglect friction in the study of the migration of oil, and I greatly regret the lack of data concerning the factors of flow for oil in small passages. The viscous lag of the heavier paraffine hydrocarbons in Dr. Day's experiments must have been due to friction. If friction ceased, there could be no viscous lag in geological migration.

Dr. Day then proposes a modification of my paper by adding another subject, the fractionation of crude oils by migration through shale, a very useful discovery of his, which, however, does not strictly concern the subject of the paper, capillary concentration. This is purely a mechanical subject and I would prefer to leave chemical and other extraneous matter for other discussions. Capillary action is independent of the nature of the capillary walls and depends only on surface tension and on pore diameter.

Fractionation by filtration is due to adsorption and to viscous lag, and these factors, especially the former, are strongly influenced by the nature of the capillary walls, but this side discussion can receive little attention here. The word "diffusion" has no rightful place in the discussion of the filtration of liquid hydrocarbons. Diffusion is a molecular process which operates only in gases and in substances dissolved in liquids. To say that since oxygen and hydrogen may be separated by diffusion, therefore diffusion may separate a mixture of two liquid hydrocarbons, violates the principles of molecular physics, unless the reference be only to the vapors of the hydrocarbons, which evidently is not the case. All the phenomena of fractionation by filtration may be explained without calling upon any

unknown or mysterious molecular process, simply by noting the controlling factors, which are: first, the selective adsorption of the unsaturated hydrocarbons, of the sulphur compounds, and possibly of others, upon the surfaces of clay particles, and secondly, the progressive frictional lag of the more viscous fractions.

I regard the first factor, adsorption, as the most important feature of filtration in nature. The effects of the second factor, viscous lag, are thought to be blotted out in the later stages of migration, when the delayed viscous elements follow the lighter ones and reach the same destination. This is indicated by the experiments in Engler's laboratory, in which the filtration was not stopped as soon as the tube of fuller's earth was saturated with oil, but the oil was allowed to continue flowing through the earth, whereupon it was observed that the succeeding later fractions became heavier.

However, when we take up the subject of fractionation, it will be necessary to consider the other alterations of crude oil, especially oxidation and polymerization, and the catalytic influence of clay; also the reasons why Arnold finds that the oil of California gets heavier as it migrates. At the first opportunity I will present my field observations in support of the essentials of Dr. Day's theory, also the chemical and geological matter concerning the modification of oil during migration and during its long storage in the sands. In this manuscript, which I left practically complete in America, I have shown that crude oil grows heavier with time, and that probably all crude oils are descended from lighter parent oils. I regard partial oxidation, polymerization, and other molecular combinations, as the controlling factors in the alteration of oil in nature.

The adsorption of the unsaturated hydrocarbons on clay, as in the experiments of Day, Gilpin, Cram, Bransky, and others, removes the more condensible elements of crude oil and renders it less susceptible to polymerization and to chemical change. For this reason, combined with the protection given by the thick bodies of shale, Pennsylvania oil is lighter than that of the limestone fields of Ohio. In both States, as well as in California, and elsewhere, the distribution of the oils of different character indicates that they are the condensed or polymerized products of oils which were originally much lighter. The viscous lag of the heavy fractions, as in Day's experiments, cannot have much geological importance, being only a temporary phenomenon.

In reply to Mr. Wheeler, he will find the first discussion of an observed pressure greatly in excess of the possible hydrostatic pressure, by consulting Orton's early description of a big gas well in New York State. Since then similar observations have been common. They appear to be especially numerous on the Apscheron Peninsula and in lower Louisiana, coastal Texas and Mexico, fields in which the communication across the

deeper strata appears to be especially open, judging from the geological structure, from the locally high temperatures and generally high pressures, from the abundance of the chemically active gas, H_2S , with some CO_2 , which has low diffusive power, and from the unfiltered character of the oil, which is asphaltic. Most of the oils are highly asphaltic and viscous, except in some of the Baku and Southern Mexico fields, where the oil contains little asphalt, but is nevertheless rich in unsaturated hydrocarbons. This is the general character of the fields along the coast of the Gulf of Mexico, which are broken by faults, dikes, volcanic necks and salt plugs. On account of these vertical passages, there is little difference between the coastal fields that develop the middle Cretaceous limestone, and those that develop sands, dolomite lenses, and fissures in the overlying shale or marl. As soon as one passes inland to the interior field of Mexico, discovered by the writer, and to the northern fields of Louisiana, which are *in strata of the same age, but unbroken*, the fields acquire an entirely different character. All of the preceding characteristics are absent, there are no great gushers nor notably high temperatures, the gas is largely methane, and the oil is of filtered type, it is light paraffine oil. Asphaltic oil and hydrogen sulphide is found only below the Eocene marl or shale, in the underlying middle Cretaceous limestone. In these interior fields the absence of strong cross fractures forced the oil and gas to filter through the shale, probably making its way largely through minute joints. Since adsorption depends mainly on the area of clay surface over which the oil moves, there is no reason why oil should not be filtered in passing through the fine joints of a thick mass of shale, although we can admit a small amount of seep through the body of the shale. In this filtration into the interior fields of Mexico and Louisiana, the unsaturated hydrocarbons were removed by the clay, leaving the oil incapable of much polymerization, and therefore it remains comparatively light. The less diffusive and chemically active gases could not pass through the shale.

With further reference to Mr. Wheeler's question, Munn and others have shown that large areas of the deep sands of West Virginia and Pennsylvania are dry. As stated by Johnson in this discussion, the dryness suggests that the pressure has forced the water out, the pressure being therefore something that opposes the hydrostatic pressure. This explanation now appears as good as any, but in the paper there are other possible explanations of the dryness of the deep sands. The very fact that the sands are dry shows that they do not contain the water required to account for the pressure, which in the gas sands approximates that of the hydrostatic pressure corresponding to the depth of the well, apparently regardless of the elevation of the outcrop. There are also some notable departures both above and below the theoretical hydrostatic pressure. Since the deep sands are largely dry, the common approximation to hydrostatic pressure can have only one explanation, which is, that the pressure is

applied through the water in the pores and joints of the overlying shales and other strata.

This in itself is proof that shale is not entirely impervious. The minute joints and pores of the shale must communicate, doubtless very imperfectly, but nevertheless with sufficient freedom to permit the transmission of pressure through their liquid contents. If pressure can be transmitted through the shale from above, it can be transmitted equally well through the rocks below. Hence there is no reason to doubt that the pressure of the abyssal fluids may be transmitted outward through the fine rock pores and joints. The work of Adams³ has shown that these passages may remain open at far greater depths than was formerly thought.

I infer a small, but locally active, very slow, outward creep of the rock fluids (liquid or gas or both) due to pressure from below. The principal reason for this inference is based on plotting the distribution of pressures in the sands of different fields of the United States and of Canada for which data are available. The data are not sufficiently complete to warrant final conclusions, but there are so many instances in which the downward increase of pressure between sands, and the increase above the surface pressure, exceed the corresponding hydrostatic head, that I have been led to believe in the very slow upward movement of water, and probably of oil and gases, across the strata; also that gas pressure is in part of abyssal origin. It seems significant that excess pressures are most common in the more fractured regions, such as Baku and the Gulf Coast, where the deep communication appears to be more open than elsewhere. Other arguments for this ascent are found in the excess temperatures in oil fields, in the distribution of oils and gases of filtered and unfiltered types, in the excess of chlorine in the associated water, and in the abundance of helium in the deep wells of Kansas.

On account of its great abundance, an atmospheric origin of the helium in Kansas gas seems out of the question. Moreover, the percentage of helium in the nitrogen of Kansas gas increases with depth, pointing toward a deeper source.⁴ The marked tendency of radium emanations to collect on clay suggests that possibly the helium may have been derived from emanations, which could more readily penetrate the rocks, but if this were its origin we would expect to find very high radio-activity, which appears not to be the case. The helium probably has risen by diffusion across the strata from a deep internal source, possibly magmatic.

The common approximation of gas pressure to hydrostatic pressure may be assigned to the fact that the weight of the water in the rock pores resists the slow outward movement, and that equilibrium is established

³ *Journal of Geology*, vol. xx, No. 2, pp. 97 to 118 (Feb.-Mar., 1912).

⁴ Cady and McFarland: *University Geological Survey of Kansas*, vol. ix (1908).

when the outward pressure is equal to this weight; *i.e.*, to the local hydrostatic head. In such cases the hydrostatic head may be regarded either as the cause of the pressure, applied in most cases downward through the overlying shale, rather than through the sand from its outcrop, or it may be regarded as a measure of the principal resistance which is met by the deep-seated pressure applied from below. Where there is equilibrium this is immaterial, but where there is departure from the theoretical hydrostatic head the latter interpretation becomes necessary. The cases of excess pressure may be assigned in general to greater freedom of communication underneath the "sand," as in the domes of the Gulf Coast, to locally more active ascent of the rock fluids from any cause, or to greater resistance above the "sand." The cases of original defect in pressure below the theoretical hydrostatic head may be assigned, in general, to the opposite causes, or to leakage, or to dryness of parts of the overlying rock. It is necessary to remember that the phenomena of oil fields are exceedingly variable, that no two fields are alike. Doubtless there are many fields in which the hydrostatic pressure of the water in the sand, below the outcrop, controls the pressure, but in the Appalachian, Rocky Mountain, and Mexican fields this explanation is generally impossible.

In this matter a word of caution is necessary against the acceptance of pressure readings with the Pitot tube as giving anything more than partial approximation to the true original pressure in the sand. The Pitot tube gives the flowing pressure at the casing head, to which must be added the loss of head (pressure) required to force the gas rapidly through the pipe and the much greater loss by friction in the sand. Since great gas wells rarely can be closed completely until some time has elapsed, it is necessary to rely on the Pitot tube for pressure measurements, but we lack data for calculating the pressure in the sand ("closed" or "rock pressure"⁵) from these measurements. The writer has failed in the attempt to make such calculations, and he will be greatly indebted to any mathematically inclined student of gas who will develop a formula or table for the purpose, which is applicable to the variable coarseness and porosity of sands.

These questions demand separate papers. In the present study pressure need be considered only in its relation to capillarity. I believe I have shown that capillarity is wholly incapable of causing pressure in the sands.

The word adsorption occurs only once in the paper and is correctly used if printed with a "d",⁶ since it refers to the moisture adsorbed on

⁵ The gas man's term "rock pressure" should be avoided in scientific discussions, since it may be taken to mean pressure exerted by the rock. "Closed pressure" is preferable.

⁶ I have not seen the paper in print, nor the proof, having only a copy of the manuscript.

clay surfaces as distinguished from the free or capillary water. Dr. Day's objection to the use of the word is not understood.

Mr. Burrell infers an upward creep of natural gas from his observations of the downward increase in the percentage of heavy hydrocarbons in the gas of different sands. This new argument tends to support my hypothesis of the outward creep of fluids, and I beg Mr. Burrell to give us this useful information in greater detail. Has he observed any downward increase in the percentage of CO₂ and of other less diffusible gases?

Gas and Oil Wells through Coal Seams

General discussion on the above subject, presented at the Pittsburgh meeting, October, 1914.

GEORGE S. RICE, Pittsburgh, Pa.—Undoubtedly there is a serious problem through the juxtaposition of gas and oil wells and coal mines, not only at the present time, but possibly of far more serious import for the future. If one examines a map of the coal fields on which is superposed the oil and gas fields of the country, one finds that they overlap to a very great extent. I may say at the beginning that we have not, so far as I know, had any very large number of accidents growing out of this situation causing loss of life. Such accidents, or rather incidents, that have occurred have all been rather potential possibilities for disasters due to the leakage of gas from wells into the adjacent coal mines. The condition has been most acute in the Fairmont district of West Virginia, and in the general coal fields southwest of Pittsburgh in Pennsylvania. The accidents which have actually occurred were summarized in *Bulletin No. 65* (Oil and Gas Wells through Workable Coal Beds) issued by the U. S. Bureau of Mines, reporting the transactions of several meetings in Pittsburgh, February, 1913, of oil and gas well representatives, State geologists, State mine inspectors, coal-mine operators, and representatives of the U. S. Bureau of Mines. This conference was partly brought about through a case in which the Pennsylvania State inspection department had brought suit to restrain a gas company from completing a well through a certain mine. The office of the Attorney General of Pennsylvania asked the opinion of the Bureau of Mines upon the matter; about the same time the Association of State Geologists, at its New Haven meeting, asked the Bureau to take up the problem. Accordingly, it was thought advisable to bring together the several interests. For the most part the danger is, I think, greater in the future than at the present time. Many parts of the coal fields of Pennsylvania, in Greene County particularly, have been penetrated by numerous wells very close together, and with the present practice of having a large coal pillar around each well, has made or will make the mining of many areas difficult or pro-

hibitive. There is not time to discuss fully what should be done, but it is believed there should be a standardization of the methods.

W. E. FOHL, Pittsburgh, Pa.—I would like to remark in connection with Mr. Rice's preliminary statement that in all these accidents, the accounts of which were collected in the bulletin issued by the Bureau of Mines, not one of them occurred without the presence of an open light. The gas which leaked into the mine from some place or other, in some way or other, always came in contact with an open flame, and it seems to me that that has a highly practical bearing upon the methods of operation of our mines. To put it in another way, it seems reasonable to suppose that if mines, in which these leakages occurred, had been worked with safety lamps, such as are used elsewhere on account of the presence of marsh gas in the coal itself, none of these accidents would have occurred.

Protecting Pillars.—It is generally conceded that oil and gas wells penetrating workable coal seams require protection from the mining operations in their vicinity, although there have been numerous instances in the McDonald field where, through lack of knowledge of well locations, all the coal has been removed from around the wells without damage to them or the miners engaged in removal.

Various proposals have been made with reference to this protection, ranging from the removal of all the coal, substituting an artificial pillar, to the leaving in place of a pillar of coal 200 ft. square, containing 0.92 acre, which in the Pittsburgh coal seam in this vicinity, having a thickness of 5 ft., would mean the irreparable loss of nearly 7,000 tons of coal for the protection of a single well.

I think it extremely doubtful that any effectual means of replacing the coal *in situ* can be found, and that it provides the best protection against the admission into the mines of deleterious gases in large volumes. It does not appear either necessary or advisable that pillars should be provided that will be absolutely proof against rupture when the final subsidence of the strata takes place after the removal of the surrounding coal. The requirement is that they shall keep the well intact during the preliminary breaks, and it must be borne in mind that these take place from the bottom upward in such manner that, when the heaviest pressure is exerted against the well and its surrounding pillar, actual mining with its accompaniment of human labor can only be in progress at a considerable distance from the well, this distance varying with the thickness of the overlying strata.

I believe there is a safe size of pillar, well within the limits of the maximum size I have quoted, for each of the mining regions in which coal, oil, and gas are being removed simultaneously; but experimental information for fixing the size of this pillar is not yet at hand.

The protection of a wall is analogous to the protection of a shaft, except that protection for the latter ordinarily comprehends support

for the more important units of the surface plant. The size of pillar actually requisite for these purposes is not susceptible of mathematical calculation, as it is impossible to forecast either the volume or the direction of the forces which will impinge on the pillar during the subsidence of the overlying strata. These can only be estimated, and the following table will show how widely authorities on mining have varied in the estimates they have made in this connection. It will be readily seen that there is no consensus of opinion among the seven authorities quoted as regards any given depth and it will also be apparent that there is no regularity of disagreement in progressing from the minimum to the maximum depths to which their formulæ have been applied in the preparation of the table.

Sizes of Shaft Pillars in Accordance with Formulæ of Various Mining Authorities for Different Depths from Surface

Depth from Surface	Feet 100	Feet 200	Feet 300	Feet 400	Feet 500	Feet 600	Feet 700	Feet 800
Length of side of square pillar								
Merivale.....	38	54	66	76	85	93	101	118
Andre.....	105	105	105	105	121	135	154	175
Wardle.....	120	120	120	130	156	180	204	231
Pamely.....	120	120	120	145	169	195	220	244
Foster.....	159	224	275	317	355	389	420	449
Dron.....	67	133	200	267	333	400	467	533
Hughes.....	100	200	300	400	500	600	700	800

In 1910, acting in conjunction with the late James Blick, sometime Inspector of Mines for the State of Pennsylvania, the writer advised the leaving of pillars for the protection of oil and gas wells as shown in the subsequent table. This table was intended for use in the mining of the Pittsburgh seam of coal in West Virginia where it had an average thickness of 7 ft. and where the extreme depth of cover was 700 ft.

Pillar Sizes Suggested by Blick and Fohl

Cover, Feet	Size of Pillar Feet Square	Area, Acres	Tonnage Re- coverable	Cover, Feet	Size of Pillar Feet Square	Area, Acres	Tonnage Re- coverable
20	40	0.04	400	300	120	0.33	3,300
30	50	0.06	600	350	125	0.36	3,600
40	60	0.08	800	400	130	0.39	3,900
50	70	0.11	1,100	450	135	0.42	4,200
60	80	0.15	1,500	500	140	0.45	4,500
100	90	0.19	1,900	550	145	0.48	4,800
150	100	0.23	2,300	600	155	0.55	5,500
200	110	0.28	2,800	650	165	0.62	6,200
250	115	0.30	3,000	700	175	0.70	7,000

Although this table was constructed only after a careful examination of the mining conditions in the region where it was intended to be applied and after a careful consideration of all the available literature on shaft pillars, it was recommended only tentatively and was accompanied by the following suggestion:

"We have in mind as a suggestion that two wells might be drilled and cased to the coal. One of these to be located near a barrier pillar, so that after the coal had all been removed up to and surrounding three sides of its supporting pillar, access could be had from the remaining side, making it possible to drive narrow headings through it to observe the crushing effect of the overlying strata. Another with its supporting pillar might be left as an island of coal in a considerable area of gob and observations taken as to the effect of removing the surrounding coal, both on the surface and on the casing of the well. The most important information, however, could be secured by a careful watch and instrumental location of all surface breaks at all of the mines of the ——— Coal Company, during the same time, with a simultaneous location of the position of the inside workings at the time breaks were made. From these it should be possible to determine to a nicety the extreme angle at which the strata may be expected to break over the solid coal and from this the size of pillar necessary could easily be established."

So far as the writer has knowledge, none of the observations here named has been made either in the West Virginia region for which they were suggested, or elsewhere; but if, as he believes, a pillar of coal in place is a necessity, it is very certain that its size can be determined only by practical experiments along the lines here suggested.

M. B. LAYTON,* Pittsburgh, Pa.—During the past few years there has been considerable discussion among coal, oil, and gas operators as to the method of drilling through coal measures, having in mind the conservation of life and property. Recently there was held in this city a conference between the coal, oil, and gas operators and the U. S. Bureau of Mines. As a result of these meetings a committee was appointed to prepare and present a bill to the State Legislature of Pennsylvania, covering what was agreed upon at this conference as the best method of drilling oil and gas wells through coal measures. The speaker was a member of this committee, and after considerable labor a bill was drafted and presented to the State Legislature, which passed first reading, and then passed into the archives of bills which never again see the light of day. Why? None of the committee knows. This was a great disappointment to the committee, as the bill as drafted by the coal, oil, and gas men was considered a safe and sane method of drilling through coal measures, having in mind the conservation of life and property. The oil and gas operators are very anxious to remove as far as possible anything that would result in the loss of life or property.

In discussing the drilling of gas and oil wells through coal measures

* Non-member.

the speaker believes it advisable to discuss the subject in the following manner: Location of well; diameter of hole and casing; the manner of protecting casing through the coal seams; and the plugging of abandoned wells.

The location of a proposed well should be determined by a survey, showing the courses and distances from two permanent points on the boundary of land upon which the proposed well is located, giving the name of adjoining tracts, township, and county. If the tract is in a locality which is being operated for coal, a copy of the description of plat should be filed with the coal operator. Should the proposed location be such that the bore hole would pass through an entry or room in the mine, the same should be re-located so as to pass through the solid coal except in case of an abandoned mine, providing it does not pass through an entry which is in use for free air to another part of the mine in operation, or a passageway for men to get to and from their work.

Where the coal is in place a hole of a diameter 4 in. greater than the inside diameter of the outside casing to be put through the coal shall be drilled at least 30 ft. below the bottom of the coal bed. Within this hole shall be placed the casing, the space between the outside of the casing and the wall of the hole to be filled with cement mortar to a height of 30 ft. above the top of the coal bed for the purpose of excluding water from the coal bed.

Where the coal is removed and the mine excavation is inaccessible, a hole of a diameter sufficiently large to permit the setting of a liner 4 in. larger in diameter than the inside diameter of the casing to be put through the coal shall be drilled at least 30 ft. below the bottom of the coal bed. Within this hole shall be placed a liner 4 in. larger than the inside diameter of the casing, and extending from the bottom of the hole to at least 30 ft. above the top of the coal bed. A string of casing shall be placed within this liner and the space between the liner and casing filled with cement mortar to the top of the liner. To exclude water, the space between the casing and the wall of the hole and immediately above the liner shall be filled for a distance of 10 ft. with cement mortar.

When the coal is removed and the mine excavation is accessible, a suitable retaining wall is laid in cement mortar about the casing. This wall shall extend from 2 ft. below the mine floor to the roof of the mine, and be of such size as to retain from 4 to 6 in. of cement mortar about the casing. This work to be completed before the well is drilled to a greater depth.

To the speaker, the greatest danger arises in the improper plugging of abandoned wells. There is no part of the work which has been carried on in such a haphazard way. Wells drilled through coal beds, upon abandonment should be filled solidly with rock sediment, sand, clay or other suitable material, from the bottom of the well to a hard and firm

stratum below the last string of casing set in above the stratum producing gas or oil. At the top of the lowest producing sand two wooden plugs at least 3 ft. in length should be placed; these to be of size or diameter of the hole and driven into place, and on top of the plugs 10 or 12 ft. of clay should be placed and thoroughly tamped down for the purpose of preventing the passage of gas, oil, or water. Immediately below the seat of each string of casing there should be driven a seasoned wooden plug, and spaces between wooden plugs should be filled solidly with rock sediment, clay, sand or other suitable material; as the casing is withdrawn, all plugs should be driven in place. In a well where the casing has been cemented in, the outer casing may be cut off at a point not less than 50 ft. above the coal bed and removed, but in any event, the hole should be filled to the surface.

The great difficulty has been a lack of co-operation between the gas and oil operators and the mine operators. *There must be a hearty co-operation and fairness* on the part of each of them if we would conserve life and property. We need a stringent law which would work no injury to either party, and pliable enough to let either party get his product out of the ground at the least possible cost. The procedure suggested does add considerably to the cost of each well, and in following it the gas and oil operator would, as far as we can see at the present time, have done all that he can to safeguard the interest of the mine operator and workmen, and they in turn should be willing to furnish complete details of their mines, so that in the making of locations the well operator would not interfere with the coal operator.

With this in mind, on the part of the gas and oil well operators, we desire to impress on you this idea: That the oil and gas well operators are willing to go to any extent that is reasonable to protect life, and remove from the oil and gas operators the accusations that some have endeavored to place on them—that they are responsible for explosions in mines, which I very seriously question. In drilling through coal measures we frequently find large bodies of gas in the coal, and after an explosion has taken place it is impossible to determine from where the gas comes.

What I have had to say to you is the result of considerable thought on the part of the coal and oil and gas men during the past two years in our meetings in preparing the bill and at Harrisburg.

GEORGE S. RICE, Pittsburgh, Pa.—There was one point I would like to hear further discussed by Mr. Layton. If I understood him correctly, he spoke as if it would be unreasonable for the coal operator not to leave a pillar of coal around a well. For the protection of life, where no other adequate means presents itself, a coal pillar should be left around the well, whether paid for by the mine operator or the gas or oil well operator. In my contact with the able representatives of the latter I have found them willing to do everything which they consider essential for safety,

but the question is, What is necessary for safety? To solve this question satisfactorily the measures adopted must be acceptable to the mine operator, the oil and gas well operator, the miners whose safety is concerned, and the people as a whole, who are concerned in conserving all the mineral resources, coal, oil, and gas. The safest thing would be not to mine the coal at all in the vicinity of gas or oil wells, or not to drill the wells in the vicinity of the mines; but this is not a practical solution, as generally the ownership of the coal, the oil and gas, and even the surface, is in separate hands, and each wants to realize on his investment. Hence the need of an orderly, systematic, and safe method of procedure, with a minimum loss of coal while obtaining the gas and oil.

Now, concerning the leaving of pillars, probably the larger the pillar, the safer the condition, but if you have holes drilled only 500 or 600 ft. apart on an average, and leave a pillar of 200 ft. square, or even 100 ft. square, you have very little coal left. If a miner does go on and mine in and around such pillars there will be enormous wastage of coal, because it is not practicable to go in and remove these separate pillars after the gas wells have been exhausted. Therefore, some reasonable compromise has to be made in this matter.

From the standpoint of safety, it is desirable to have the pillars as large as possible, but in the interest of the conservation of resources it is desirable to have them as small as possible. Somewhere between, there must be a compromise.

M. B. LAYTON.—Mr. Rice seems to get the wrong impression of this idea of drilling through coal measures. When the coal has been taken out, the surface breaks down, and all that the gas and oil well operators want is enough coal or concrete around the casing to protect it from sulphur water. The life of the ordinary gas and oil well is short, and we are certain of the result which will eventually come in the district in which the coal is mined, as to the sinking and breaking down of the strata covering coal measures. If the territory is drilled before the coal is taken out and the casing in the well has been protected as suggested, and in case of abandoned wells properly plugged, there will be no danger of gas escaping into the mine, and you have no right to ask the gas and oil well operator to buy the coal in order to get at the product from below, when he is willing to spend his money in doing everything that is possible for him to do to safeguard and confine the gas in the casing, and in the case of abandoned wells to the gas-bearing sands or strata.

GEORGE S. RICE.—Buying the coal would not meet the proposition of conservation.

M. B. LAYTON.—Conservation of life is the important thing to us, and I desire to state most emphatically that the greatest danger from gas

escaping into coal measures is from the improper plugging of abandoned gas and oil wells.

R. D. HALL, New York, N. Y.—It seems to me that possibly the gas men and the oil men are partly to blame for the difficulties they have with the coal owners because of their desire to drill a great number of wells. The mining operators could possibly manage to continue mining around gas wells without marked danger if the wells were not so frequent. The coal men could either leave big pillars to protect them, if pillars are desirable, or arrange to take the coal out around them in such manner as to do the least detriment to the wells. But the oil and gas men have drilled holes with such extreme frequency that in certain districts it is almost impossible to make any arrangements at all to safeguard the wells. The gas and oil men seem to have much faith in the strength of the little concreted pipe stems which they are putting down through the earth. If they had had more experience in mining they would realize what terrible catastrophes sometimes occur in mines, and how large an area will sometimes cave in at one time, and they would realize the titanic forces with which their little pipe stems are endeavoring to cope.

A little while ago, in Oklahoma, there was a squeeze which suddenly shut in 13 men. They have never been recovered. The cover of the mine at the deepest point was about 700 ft. It was a huge working with about 15 levels. All of a sudden the mine closed in almost all over. When you are dealing with forces of such magnitude, it is hard to arrange any protection at all for the pipe lines. Pillars 100 ft. through, and even more, have been crushed down with a general movement of the mine. It seems that the oil and gas men ought not to put down so many holes. Indeed, it appears unnecessary that there should be so many.

It also is apparent that the mining authorities ought to regulate their work in such a way that there would be the least danger when the falls take place. There are parts of England to-day where coal is being mined all around the shaft. No shaft pillar is left, the operators depending on the symmetrical falling of the rock to prevent any distortion of the cageway. And as a result of these precautions there is very little of such distortion; in fact, it is doubtful whether just as good results cannot be obtained without a shaft pillar as with one. So with proper arrangements it may be safer to take out all the coal around the bore hole than to leave a pillar, unless that pillar is so immense as to be unquestionably adequate. For it must be remembered that the strain does not cease at the edge of the pillar, but extends well over it. That has been recognized for years, and that action has been said to be due to the "draw." It can be very easily seen how logical that is, because when we build continuous bridges which pass over several abutments we have to put tension members on the top of the bridge over the abutments, or there will be a breakage of the bridge on the top of the abutment, despite

the fact that there is a support at that point. And the strain those tension members have to sustain is considerable. So it is true that there is a tensional stress which is important, even where the pillars are extremely wide. It is in providing for this stress of the measures above the coal pillar that we find our perpetual risk. It is questionable to my mind whether it would not pay the operator to take out all the coal around the pipe, meantime taking precaution to make the workings symmetrical around the bore hole, but that cannot be done as long as the oil and gas men persist in putting down so many holes.

DAVID T. DAY, Washington, D. C.—Mr. Hall speaks as though he was laboring under the idea held by some people who are not in the oil and gas business, that they drill gas and oil wells for fun. It is very costly fun, and I doubt whether in these days, particularly in Pennsylvania and West Virginia, any oil or gas hole is drilled on a man's own property, which it is not absolutely necessary to drill. In an anthracite mine gas frequently comes through the coal so rapidly as to make a singing noise. The coal does not hold the gas back at all, apparently. What is the use of leaving any pillar walls? It is of no use in holding back the gas. One inch of cement will hold better than any 10 ft. of coal column. The very important feature was brought out by Mr. Hall, of the chances of the pipe and cement collapsing, unless it is excessively thick, where collapsing of all sorts is taking place in the mine. These are matters which must be given very careful consideration, especially where the chances of deflecting that pipe by roof falls, etc., in the mine are imminent. These may be greater in other mines than they are in the mines of Pennsylvania and West Virginia, but still they must be considered. If Mr. Rice will give us the plans which he has in view for substituting columns of some sort that will keep the pipes from breaking, that will help the subject considerably.

GEORGE S. RICE.—There is hardly time to discuss that matter because it involves questions of mining and looks, possibly, a little too much to the future. I am strongly convinced that the hydraulic method of stowing the workings, not filling afterward as practiced in the anthracite district, but as the face advances, in the way now practiced in many European mines, is one that we will come to ultimately in this country. If such hydraulic filling is done, we know, from very positive data that we have, that with the proper filling material we can keep the total subsidence down to 5 to 10 per cent. of the thickness of the excavation, and the overlying strata goes down so evenly that it is not ruptured. I was much interested a couple of years ago in visiting a mine under the famous Krupp works in Germany, where the government had previously forbidden the Krupps to take out the coal, which they owned, under a certain million dollar building because it was manufacturing ordnance; but when this system of hydraulic filling had proved itself so effectual

elsewhere they were permitted to mine out all the coal, and it was taken out without any trouble whatever to the building, through the employment of hydraulic stowing. If you can go under a great building of that kind with its foundations and all its elaborate machinery, it certainly would not be very difficult with comparatively flexible pipe arrangements to take up any small subsidence which might occur. Therefore, I think we can look forward to a time when we can mine right along without regard to drill holes; but that is looking entirely to the future, because it will add to the cost of mining. I cannot see that it can be done at the present time. However, I do think that, even at the present time, in some instances in the course of pulling the mine pillars where the coal bed lies horizontally, it may be possible to remove the pillar around a well by substituting a careful packing of rock and refuse, and arranging the casing with a slip joint at the bottom of the mine workings, the inner casing or tubes carrying the gas being free or unattached to the outer casing. With this arrangement, described¹ at the conference of 1913, the outer casing, which is gripped by the surrounding strata, would be carried down with it gradually as the packing is compressed, without disturbing the inner tubes, which will rest on the strata underlying the coal bed. The packing in such cases would need to be carefully done, with special reference to the general withdrawal of the neighboring pillars, to obtain a regular and vertical subsidence. However, even if there was some side movement higher up, the casing and the free inner tubing would probably be only bent and not ruptured, and, with the maintenance of a free vent to the atmosphere between the outer casing, or casings, and the inner gas tubes, there would seem to be no danger of gas leakage into the mine.

R. D. HALL.—I think the remarks relative to gas and oil wells not being drilled for fun possibly overlooked certain facts. The main reason for drilling so many holes is to make the fields produce large quantities of gas and oil in as short a time as possible, with the idea of producing the greatest profit from each field with the least amount of expense, which is a perfectly natural desire. But on the strength of that anxiety of gas and oil men the coal operators have been obliged to leave pillars and hunt around for pieces not so drilled. Thus their mining work has been delayed and their profits reduced. I may here add that these profits are very much less than the gas and oil men are getting. Another reason why the oil and gas men put down so many holes is that their territory is pretty thoroughly broken up. They do not buy contiguous and continuous territory, and as a result are trying to drain one another's property by their holes. And this method of operation is not really advantageous to the public generally.

¹ *Bulletin No. 65, U. S. Bureau of Mines*, pp. 15 to 18 (1913).

O. P. HOOD, Pittsburgh, Pa.—It has been my opportunity during the last year to observe the operations in Oklahoma, and see what changes might be desired in the regulations drawn up a year ago. I think I may say that we feel like placing more dependence on mud and less dependence on cement, wherever it is used. In plugging wells, we are growing a little surer every day that the substitution of clay or mud for wood plugs or cement plugs is the proper thing, and that the protecting of pipe lines by something put between casings, as Mr. Layton suggests, is better done with plain mud than it is with cement.

G. A. BURRELL,* Pittsburgh, Pa.—Our natural gases usually contain other combustible constituents than methane. The paraffin hydrocarbons, other than methane, amount to about 14 per cent. in natural gas, whereas in normal mine air methane is practically the only combustible constituent, so that by a careful chemical examination of the mine air the presence of natural gas might be detected in time to avert a large inrush.

DAVID T. DAY.—It has been stated that no mine accident, due to the gas escaping from gas wells into the mines, has occurred except where there were open lights. I would like to know whether the analyses of the mine air, as far as they have been available in such cases, have indicated a tremendous inrush of gas entirely out of proportion to the proportion of natural gas found in anthracite mines or under normal conditions in gaseous mines, or whether the same condition practically exists in both stages.

G. A. BURRELL.—The quantity of methane that comes from some anthracite mines per 24 hr. is equivalent to that coming from some gas wells.

GEORGE S. RICE.—I might bring to mind the one especially historic case of the Fairmont mine in West Virginia, where a certain well, which had not been commercially successful, was capped and in some way the gas pressure became equalized through the different casings. Inner casings had been used, but all the casings were covered by a large cap over the outer casing and the openings closed. The pressure was said to have been about 1,000 lb. per square inch. The well passed through a mine (No. 47) but was surrounded by a large pillar, about 125 by 380 ft. in area. There was another mine (No. 49) adjoining, separated from the first mine by a barrier pillar about 75 to 100 ft. in width. On a Saturday or Sunday night when the mines were not in operation and the fans had been shut down, the mines being considered non-gaseous, explosions occurred in each of the two adjoining mines, due to the men going in with open lights and igniting gas. Fortunately there were but few men in the mines, and the mines were very well protected from dust

* Non-member.

explosions through efficient humidifying, so that the explosions were local and only three men were killed. It was quite freely admitted what the conditions were, and the details have been published in a paper by one of the coal operator's engineers in the *Transactions of the West Virginia Mining Institute* for 1911. It was found after the explosion that there were a series of gas blowers issuing from the floor for a distance of about 2,300 ft. in a diagonal line extending across these pillars. It is supposed that there may be a line of structural weakness through the strata, and at some deeper point gas leaking from the well under this high pressure had burst up through the floor of the mine. When the cap over the casings was taken off these blowers ceased. Samples were taken from the blowers, the analysis of which showed it to be natural gas, and not mine firedamp or methane. But it is easy to see from this case that if you have a leakage of such a nature, a more serious disaster might result.

DAVID T. DAY.—It is hard to conceive of a well being kept closed for commercial reasons, with so high a pressure.

W. E. FOHL.—The well had not been abandoned. It was simply inconvenient to use the gas on account of its being distant from pipe lines. The capping of it was a mistake. It had an outlet, but by mistake of a man in the field that outlet was closed. The pressure, as it was told to me, was 900 lb.

M. B. LAYTON.—We frequently abandon a well in a field with a rock pressure of several hundred pounds, but the volume of gas is so small that it does not pay to operate the well.

GEORGE S. RICE.—While a well may produce too small an amount of gas to be commercially of value, if it is inclosed, or filled in merely at the top, as has sometimes been done, instead of filling from the bottom, it presents a serious menace to a mine. Though the daily flow may be small, or even insignificant, if the pressure is high a large amount may be stored under a pressure of many atmospheres; then if tapped in the mine, it will rush out, and when mixed with 18 times its volume of air makes a great body of explosive gas with tremendous possibilities for danger to life. I call to mind two instances where uncharted, abandoned wells, of which there were no traces on the surface, were encountered in mining; in one of these instances the casing through the coal was still in place when struck by the mining machine; as the casing was not broken, the leakage was not sufficient to be serious. In the other instance the casing had been pulled and the gas came out in volumes and was ignited. Fortunately it was ignited before it became mixed with the air, so it merely burned. While this caused a serious mine fire which involved closing the mine for a period, the drill hole was located by surveying and probing, and was then redrilled, cased from below the coal, and vented to

the surface. The mine fire was finally smothered by the erection of underground stoppings; but there was serious loss of output for the operator, and loss of wages for the miners, which would have paid many times over for original complete filling of the well.

Another lesson to be drawn from this is that there should be a law compelling the public filing of a map to show the exact location of every well, together with a complete record showing the depth of hole, the character and position of all casing left in the hole, and of the plugging or filling. Our State geologists would like even more information about the wells made a matter of record; but there might be some argument raised against giving complete records of the strata, and the gas, oil or other mineral found, but surely there can be no serious objection to the records needed for the safety of future mining, not only in the upper coals or other mineral beds, but also beds at lower horizons not now commercially workable.

GEORGE S. RICE.—I think those who were not present at the conference a year ago, realize the amount of work that has been done and the amount of material threshed over before the suggested laws were formulated. It seems to me the greatest danger is from small wells, and those are the ones which frequently have the highest rock pressure. That is the reason it is necessary, in plugging wells, to fill them solid from the bottom to the top, so there is no chamber where gas could accumulate. I think most of the accidents, outside of the one at Fairmont, have occurred in the striking of wells, and the sudden liberation of a small pocket of gas. That is the danger in the old abandoned wells, and that is the object of these specifications for filling wells with soft material from bottom to top.

Another thing that should be mentioned in connection with this is the mud-laden process of drilling wells. I know that certain rather large oil and gas interests think that the drilling of wells by the mud-laden process will eliminate all troubles. Now it is expressly stated by the bulletin on this subject that to utilize the gas which is shut off in this process, you simply pump in the head of water, and the rock is filled with the sediment that is deposited automatically. The sediment in this process is clay, largely colloidal in character. And that condition will be maintained by the extraction of salts from the well walls. I think we can all conceive conditions in which the head of water would not be maintained, and the sand would be automatically opened up.

Salt Making by Solar Evaporation

Discussion of the paper of W. C. PHALEN, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2249 to 2265.

DAVID T. DAY, Washington, D. C.—Consider the United States broadly and the distribution of the salt industry. Think of the old solar-evaporation process up in New York State, then of the more recent processes in Michigan and Kansas and Louisiana, where the rock salt furnishes a large industry, then of the solar-evaporation process at Salt Lake, then turn to the crucial place in the United States, where there is an embarrassment of salt, and that is California. I am not forgetting the immense body of salt east of that in the Virgin River country of Utah; but in California the situation is interesting. There are very large quantities of salt in the Searles Marsh region; again, in Southern California, at the southern end of Death Valley, there are two or three mountains of salt which have been explored recently, where the salt is sufficiently pure to be used at once. There is also a large deposit of gypsum, and it would be possible there, by extremely cheap means, to produce an amount of salt sufficient to furnish the United States' greatest needs. The principal reason that capital has been deterred from developing such a large mass is not the fact that solar evaporation produces a great deal of salt around San Francisco, but the fact that if potash salt is to be produced in connection with the salt in the borax region a great deal of ordinary salt must be disposed of. It cannot be dumped, for it would get back into the brine again; the first rains would bring that salt back into solution again. It must be sold, given away, or hauled to a distance, which would mean a considerable freight charge. The salt will be going begging.

E. G. SPILSBURY, New York, N. Y.—The remark of Dr. Day's regarding the likelihood of the production of a large amount of waste salt brings to my mind the condition that obtains in Germany in all of the potash fields.

In Germany salt is a government monopoly. In the development of the potash deposits, it becomes a grave question as to how much salt they may be allowed to extract and bring to the surface.

In sinking a shaft that I was connected with, near Hanover, we had to go through 800 ft. of rock salt before we struck the potash deposit, which always, in that section, lies between what is known as the "Old Salt" and the "Young Salt." In this case the Young Salt was about 800 ft. in thickness, and we had to go through that. The question of what to do with that salt was a serious one. The government insisted that

we build a covered bonded warehouse for it and all of the salt taken out of that 21-ft. diameter shaft for 800 ft. in depth had to be stored in the warehouse and kept in bond for the government. The suggestion that we should merely stack the salt up was not allowed, because in rainy weather the washings from it might probably be collected by the farmers around and used for pickling, and so the government would lose the royalty on it.

Of course, after we got down into the potash deposits the trouble was not as great, and to-day those bonded warehouses are all empty. The salt has been taken back into the mine and used for filling where the potash has been taken out. That Dr. Day should have pointed out the real difficulty of potash recovery is very interesting.

Another matter in this paper which attracts my attention is the difference of the methods used in this country for the concentration of the brines from those used in Germany, Austria, and Hungary. Instead of using "aprons," as described in the paper, they erect a long staging, about 30 ft. high and varying from one to two or three miles in length. Both sides of the staging are filled in with brushwood, with the fine ends down. The brine pumped from the wells is conveyed along a perforated trough the whole length of these evaporating and concentrating stacks, and trickles down on both sides of the 30-ft. high brush heaps, and in that way the evaporation is carried on very cheaply, profitably, and perfectly. The droppings of concentrated brine are collected in troughs in the bottom, and conveyed to the salt-boiling establishments. It seems to me that in this country, particularly in the West, an adaptation of this method would be much cheaper than the methods now used, the evaporation in ponds, which necessitates the plowing up of the salt and the rehandling—when the concentration can be brought to the actual point of saturation by such a cheap and labor-saving contrivance as they have in Germany.

W. LINDGREN, Boston, Mass.—I simply want to call attention to the fact that in the German schools a great deal more time is given to the subject of the treatment of salt than in the United States. You will recall, all of you, the title of the Prussian journal, *Zeitschrift für Berg Hütten und Salinenwesen*, which is studied extensively, and in all the technical mining schools there are special courses on this subject.

DAVID T. DAY.—It seems to me that Mr. Spilsbury's proposition is one worth a good deal of study and consideration. The lifting of all this salt while it is in solution, by a simple pumping operation, means avoiding all shoveling, and manual labor, which is pretty hard to get in the neighborhood of San Francisco. To take this salt up in a saturated state seems to be an ingenious idea that ought to be developed.

E. G. SPILSBURY.—The question of pumping the entire amount of sea water up to 30 ft. and distributing it as described would not be as eco-

nomical as the first operations in the present salt pits or tanks. In Germany, their method has been developed, not for handling sea salt, but for pumping brine from wells. The brine is pumped from wells, and therefore the extra 30 ft. of pumping did not amount to very much. If it were applied here to the American conditions, I think it would be more economical to make the first concentration in the solar pits and then pump the fairly concentrated brine up and finish the concentration on the brush evaporators.

H. RIES, Ithaca, N. Y.—Is that German method still as extensively used in Germany as heretofore?

E. G. SPILSBURY.—Yes, you can notice these long high brush heaps for miles along the railroads, in the salt-well country.

H. RIES.—Does the bittern rejected at San Francisco Bay contain any bromine?

DAVID T. DAY.—Yes, the amount of bromine seems to be considerable, but practically no use is made of it, except an extremely small amount for medicinal purposes.

Asbestos in Southern Quebec

Discussion of the paper of JOHN A. DRESSER, presented at the Pittsburgh meeting, October, 1914 and printed in *Bulletin* No 93, September, 1914, pp. 2267 to 2274.

E. G. SPILSBURY, New York, N. Y.—Referring to the methods of crushing, is there any special way of feeding the material—the cross-grained asbestos—to the rolls so that the crushing effect is only lengthways of the fiber, or does it all have to go in and produce its portion of broken fiber?

JOHN A. DRESSER, Sault Ste. Marie, Ont.—Mr. Spilsbury's question is a very important one, economically. Unfortunately, it is not possible to feed the crushers in any way such that the fiber may not be broken and shortened, and its value much reduced. The coarse crushing is usually done by jaw crushers, after which gyratory crushers are used for the finer crushing. Rolls are also used for the finer crushing. Where rolls are used it is necessary to have a teasing apparatus for bringing out the fiber, which is apt to become matted by the action of the rolls. Any hard material in the rolls is apt to break and shorten the fiber. If we have asbestos fiber $\frac{5}{8}$ in. long that might be worth \$100 a ton, and that fiber is broken into two parts, it might only be worth about \$30 to \$40 a ton. Some mills have attempted to do without rolls entirely and are doing so. After the fiber to about $\frac{1}{4}$ in. has been removed, the material is fed to an especially designed machine known as the "cyclone" crusher.

It is said to be a development of the principle of the fanning mills used in the West for blowing the husks off grain. It consists of steel chambers in which are two fans made like screw propellers, which usually face each other and are carried on horizontal shafts. These weigh 90 to 100 lb., and revolve at about 2,000 rev. per minute, in opposite directions, and the material fed in is finely pulverized. The smallest particles of asbestos are thus released. When the material reaches that stage it means almost the destruction of the fiber; so the practice is to draw off the fiber wherever it is possible to do so. But there seems to be no means of doing this completely without seriously lessening the value of the product by shortening the fiber.

DAVID T. DAY, Washington, D. C.—What is the material known as asbestic?

JOHN A. DRESSER.—An ordinary mortar carrying a certain percentage of very short fiber, used as a binder, and possibly because it has a little better fire-resisting qualities than ordinary mortar. The fine material which is added to the lime is hardly distinguishable from dust. It is said to give slightly better fire-resisting qualities, but I do not know that its binding qualities are especially important. It gives a very smooth surface. It is a by-product at one of the larger mines.

D. T. DAY.—Is there any effort made to use the serpentine which invariably occurs on either side of the cross-fiber vein?

J. A. DRESSER.—No, except in so far as the tailings are used for asbestic.

D. T. DAY.—What has become of the deposit of asbestos in the bottom of the Bright Angel trail in the Grand Canyon.

W. LINDGREN, Boston, Mass.—It is still there, I have no doubt, but the question is to get it away from there.

I have a couple of questions I would like to ask Mr. Dresser. We all know what exhaustive examinations he has made in the asbestos regions, so I would be glad to have him give the results of his examination regarding these queries: In speaking of the slip-fiber variety of asbestos, does the transformation of the rock to this material go through the serpentine stage or is it directly altered to asbestos from pyroxene or amphibole? Another question concerns the gravitative differentiation of stocks into acidic and basic rocks. We know that such separation takes place in sills, but is it not a little difficult to apply this theory to stocks? In stocks the separation usually takes place along the periphery, the basic material seemingly consolidating first along the margins.

J. A. DRESSER.—In so far as is known from a general examination, I think that the alteration in the first instance has been to serpentine in each case. In the case of slip-fiber asbestos deposits, however, the

serpentine is principally derived from pyroxene, but this pyroxene is probably primary.

As to the question of differentiation of stocks, there is a variation vertically as well as horizontally. We find stocks that have an entire ring-like arrangement, though they are seldom symmetrical. The basic core is frequently found near one side of the stock in this annular arrangement. We get this in all states from a complete annular arrangement to an arrangement going three-fourths way around and finally passing out into a laccolith, or thickened sill. Differentiation seems to have taken place within the stocks themselves as there is an acid cap forming the upper parts of many of them. So we have cases of hills showing serpentine near the base but covered entirely over the top by diabase or porphyrite.

In one case a man, who thought that mining meant always driving a drift or shaft, drifted into diabase at the side of a hill, passed through diabase, pyroxenite, and finally got into peridotite, in a distance of scarcely 100 ft. Peridotite, pyroxenite, and diabase are all differentiations from a single magma and arranged in this order from the base upward and from the center outward. This arrangement, which is well established for this district, is comparable to that described recently by Duparc and Pamfils in the Ural district in Russia.

H. A. WHEELER, St. Louis, Mo.—Some years ago a deposit of asbestos was discovered in the serpentine hills at Tompkinsville, Staten Island, N. Y. It occurred in fibers 10 to 20 in. in length, that were very coarse and brittle. After some desultory prospecting it was finally abandoned, as it occurred in small pockets, not veins.

H. RIES.—What is the greatest depth recorded at which asbestos is found? You said pits had reached the depth of 200 ft., I believe.

J. A. DRESSER.—As far as I know 325 ft. is the maximum depth of the largest pit. Borings are reported from which some little fiber was obtained at 400 ft.; but I do not know in what quantity. There appears to be little difference in the quality at different depths. It is always irregularly distributed, making the ground extremely spotted. Because we have it at 300 ft. in some places it does not necessarily occur at that depth in another place. I should say that is the greatest depth that it is undoubtedly proved. We are usually able to work the pits all winter. At one plant there are some underground workings of considerable extent. Ordinarily there is a good deal of loss from snowfall if the output is not properly managed, but under ordinary market conditions all can be worked the year around. The rain causes a serious hindrance in workings, as well as snow.

Quarrying Shale by the Tunnel System

Discussion of the paper of DWIGHT T. FARNHAM, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2357 to 2364.

DAVID T. DAY, Washington, D. C.—Do you know anything about the character of that clay as to organic matter?

H. RIES, Ithaca, N. Y.—No; it is a dark brownish clay, sandy in its nature. The brownish coloring is probably due to organic matter. Some lignite is also present.

DONALD W. ROSS.*—I have worked at the plant under consideration. I have not seen Mr. Farnham's paper; but in answering the question as to the organic matter, I might say there is practically no organic material scattered through the shale. There are, however, several layers of carbonaceous shale and rather low-grade coal interbedded with the formation.

There are two very serious drawbacks—one which threatens to shut down the plant and another which merely means the expenditure of money. The first is taking care of the sediment from the washing away of the overburden. The government is doing extensive work at different points about Lake Washington with a view to creating a fresh-water harbor for Seattle. I understand that they keep at the Denny-Renton people to keep the channel of Cedar River clear. The Denny-Renton Co. can hardly help filling up the channel so long as they continue their hydraulic work, so this is probably one of their most serious problems. The other is this: There is a large percentage of what I should call micaceous shale or sandstone—a sandstone which contains a large amount of light-colored mica. Of course only a limited amount of this can be used in the manufacture of paving brick. As they shoot down the bank after the overburden of gravel and loam is washed away, these large boulders of micaceous sandstone come down. The strata have apparently been crushed where the clay pit is located so that the exact formation cannot be readily determined. These pieces of sandstone, 5 or 6 ft. in diameter, have to be blasted and removed from the shale. They bother both in the open pit and in the tunnel mining. In the tunnel mining of the shale blocks frequently cover the holes in the roof of the tunnel through which the shale is loaded into cars.

DAVID T. DAY.—About how many tons of that bituminous shale do they get in a day?

DONALD W. ROSS.—I believe, owing to a disagreement with the

* Non-member.

miners, that they have dug none of this coal during the last three years. The coal is of rather low grade, compared with other coals of the region, and is taken from a vein some 3 or 4 ft. thick, which dips approximately 15° to 30° to the southeast. The other coal vein is controlled by the Seattle Electric Co., and is approximately parallel to this vein and about 600 ft. below it. Dr. Fettke tells me that it approximates 15 per cent. ash. I have seen similar coals from other mines that contained as high as 60 per cent. ash.

H. RIES.—That coal you speak of is not in the main bank?

DONALD W. ROSS.—The entrance to this Denny-Renton coal mine is situated about 300 ft. north of the shale pit toward the town of Renton. I am inclined to believe that the coal vein, due to its dip, passes beneath the floor level of the pit.

The Occurrence, Preparation and Use of Magnesite

Discussion of the paper of L. C. MORGANROTH, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2345 to 2352.

D. T. DAY, Washington, D. C.—I would like to hear some further detail as to the California deposits.

L. C. MORGANROTH, Pittsburgh, Pa.—We have examined the California deposits at various times; I have been over practically all of them myself. These deposits are all of the massive variety. While it is possible that it could be mined and used for metallurgical purposes, it is not feasible because it costs too much. At the present time magnesite which is mined in California is used in the making of plaster, or by the paper trade. Careful figures show that if the California magnesite could be delivered to Atlantic seaports in the crude state at not over \$4 a gross ton, and assuming that it would give the same results as the Austro-Hungarian magnesite, it would be cheaper to use the latter. It is also more difficult to dead-burn the massive magnesite, for while it starts to give up its gas readily, the difficulty consists in driving off the last few per cent., and unless this is driven off the magnesite will absorb CO_2 from the air, so that in a few months it may have 10 or 12 per cent.

DAVID T. DAY.—How is the Austro-Hungarian supply at the present time?

L. C. MORGANROTH.—There is little scarcity at this time. There was a large stock in this country at the time the war broke out. Due to the curtailment of the iron and steel industry, many of the largest steel companies have a sufficient amount of magnesite to last them a year at

the present rate. It is also possible to get magnesite at the present time; in fact, several cargoes are expected in before the first of the year. There will be an increase in the price, due to a much higher ocean rate.

DAVID T. DAY.—How thick are the Austrian deposits? What preparation does it receive at the quarry?

L. C. MORGANROTH.—Some of these deposits are very large, several containing over 5,000,000 tons and one probably twice this amount.

The magnesite as it occurs is mixed with dolomite veins and sometimes with quartz veins. The labor at the quarry consists in picking these impurities out, the magnesite being broken down to small lumps and all pieces of dolomite or silica adhering to same chipped off. There is always a certain amount of dolomite that cannot be separated in this way, but after the magnesite is burned it shows lighter against the magnesite and this is picked out by hand. Within recent years magnetic separators are being used, which are apparently working very satisfactorily.

DAVID T. DAY.—How thick are the seams in California and how do they compare with the deposits in Greece?

L. C. MORGANROTH.—The California deposits do not compare with the deposits in Greece as to size. Most of the California deposits are in very thin veins, some being but several inches in thickness, and while they may run into a mass of about a ton, the majority of them are even smaller. There is one comparatively large deposit near San Francisco. It is lens-shaped, probably 40 or 50 ft. in thickness, with its length undetermined. However, it is 40 miles from the nearest railway station, where the material was being delivered by automobile trucks. At the time of my visit, several years ago, it was not being operated, and I do not believe it has been operated since. The stockwork variety of magnesite is also found in Venezuela and in Mexico. Some of the deposits in Greece are very thick. One noted was 50 to 60 ft. in thickness, 150 ft. high and several hundred feet in length. If there was a market for the massive variety of magnesite along the Atlantic seaboard it would be cheaper to bring it from Greece than from California.

DAVID T. DAY.—A great deal has been imported from Greece, has it not?

L. C. MORGANROTH.—At one time some amounts of the crude Grecian magnesite were brought to this country for use in the manufacture of CO₂ gas, but I doubt if the importations for any one year ever reached 10,000 tons. At the present time all that is received comes in the caustic state.

E. G. SPILSBURY, New York, N. Y.—What is the cost of transportation from the mines in Austria-Hungary to the seaport?

L. C. MORGANROTH.—The cost varies, depending on the location of the quarry, ranging from \$2 to \$3 per metric ton.

E. G. SPILSBURY.—What is the cost in California of transporting it to the railroad or to the seaport?

L. C. MORGANROTH.—I cannot give any exact figures on this, but from inquiries made believe the lowest freight rate from any good deposit to a seaport would be \$2. The rate of ocean shipment from a Pacific seaport through the Panama Canal to an Atlantic seaport would probably be \$5 to \$6 a ton. If there was use for the California magnesite, the cost of delivering Grecian would be considerably less, as the quarry is located within three miles of the coast, where the calcining kilns are located, the magnesite being delivered to the kilns by an aerial tram. The ocean rate from Greece to Atlantic seaports ranges from 10s. to 12s. a ton.

DAVID T. DAY.—If this deposit near San Francisco were nearer the railroad, would there be any hope for that?

L. C. MORGANROTH.—Very little; for in addition to the advantage that the Grecian would have over it with no railroad freight rate and lower ocean rate, labor is much cheaper there.

E. G. SPILSBURY.—What is the practical difference in chemical composition between the California and the Grecian variety?

L. C. MORGANROTH.—Practically none.

DAVID T. DAY.—Is the California variety near San Francisco anything like as pure as the Grecian?

L. C. MORGANROTH.—Yes, but the veins are thinner.

DAVID T. DAY.—But it is thicker than the Swedish?

L. C. MORGANROTH.—Yes. The Swedish, as I have stated, is a distinct variety, but it is very impure, being very intimately mixed with serpentine. Probably 60 per cent. of the vein as mined is waste.

The Appraisal of Coal Land for Taxation

Discussion of the paper of H. M. CHANCE, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 91, July, 1914, pp. 1461 to 1466.

R. V. NORRIS, Wilkes-Barre, Pa.—In his *résumé* of coal-taxation methods Mr. Chance has apparently confined his investigation to bituminous coal, so a summary of the anthracite situation as to coal assessment should be of interest.

The anthracite region of Pennsylvania in its 484 square miles of measures lies in the counties of Wayne, Lackawanna, Luzerne, Carbon, Schuylkill, Northumberland, and Dauphin, and in each of these different methods of assessment have been adopted.

The assessments for taxation in Pennsylvania are made under Acts of the Legislatures of 1841 and 1842, under which the assessors are required to "assess rate and value every subject of taxation according to the actual value thereof, and at such rates and prices as the same would bring at a *bona fide* sale after due notice." Under this general law coal land has been assessed as such, but up to about 25 years ago the coal was assessed at rates so low as to be unimportant, though the rates in each township and municipality, even in the same county, showed no agreement or ascertainable basis.

In the late '80s, the need for greater revenue drew attention to the possibilities of collecting much larger taxes from the coal lands and, spurred by public demand, the assessing authorities in some counties undertook a reassessment of coal. The method pursued was to demand from each owner a statement of the area and average remaining thickness of coal on his lands. On this return, made in some cases under oath, an assessment based on foot-acreage was levied, with a resulting great increase in valuation, but the valuation per foot-acre was low and no serious objection was made. This assessment with occasional increases of rate remained in force until 1907, when, in the triennial assessment of that year, a complete revision of the valuation was again attempted, engineers were employed by the counties to check the accuracy of the returns from the owners, and, based on the foot-acreage thus obtained, an enormous increase, to \$65 per foot-acre, was made in the valuation. This assessment was resisted in the courts by appeals resulting in very extensive litigation, and a final determination of the validity of the 1907 and all subsequent assessments has not yet been obtained. In the first appeals tried the rate calculated on a foot-acre valuation was reduced by the Luzerne County court to about \$45, while the Lackawanna County courts reduced a larger rate to \$67. The county authorities were, how-

ever, dissatisfied with these results and have since tried other appeals from the same 1907 assessment, but these latter suits have not yet been decided even by the county courts.

Since the guess valuation of a generation ago was given up, four methods of assessment have been attempted or suggested.

First: Valuations based on actual sales.

Second: Valuations based on foot-acres of coal remaining in the ground or remaining available.

Third: Valuations based on royalty values.

Fourth: Valuations based on capitalized estimated profits.

Of these methods, the first, valuations based on actual sales, is indicated as the only strictly legal method by various Pennsylvania Supreme Court decisions. Of the foot-acre method, in Reading Appeal (Pennsylvania Supreme Court Reports No. 229, p. 465), the court says "that the foot-acre value for ascertaining valuation of coal lands of the appellant for the purpose of taxation is not a proper measure of their value;" while the court disposes of valuations based on royalty, and by implication those based on profits (Pennsylvania Supreme Court Reports No. 299, p. 470):

"Its market value is its fair selling value for cash, not payable as royalty strung out through a long series of years, but payable at the time or as soon thereafter as the value could be determined. Such a method does not make allowance in undeveloped territory for the length of time coal may lie in the ground unmined, undeveloped and unprofitable. It is impossible to reduce to a scientific basis and to mathematical precision the elements of value entering into the present selling price of a tract of coal land. The question is not what earning power coal lands may develop in the future, but what they are actually worth in the market at present."

The sales method has been used both in justifying and in attacking the 1907 anthracite valuations and has resulted in the introduction into evidence of a large number of "sales" of coal land, generally the coal under very small tracts, with but few sales of acreages large enough for operation, and none of this size except sales of operating collieries where the value of the lands could not readily be separated from the value of the improvements and colliery plant. The sales shown have indicated values from \$200 or \$300 up to \$10,000 per acre, with an average for the best lands containing beds aggregating 60 ft. of coal of about \$3,000 per acre or \$50 per foot-acre, equivalent to about 5c. per ton for the coal available for shipment, estimated at 1,000 tons per foot-acre.

In the face of this, the 1913 assessment has been raised throughout the anthracite region. The rates determined by a foot-acre calculation, but returned in gross price per acre to evade the Supreme Court ruling, are as follows:

	Per Foot-Acre
Wayne County.....	\$67
Lackawanna County.....	175
City of Scranton.....	300
Luzerne County.....	250
Northumberland County.....	48 to 61
Schuylkill County.....	9.84 to 68.52
Dauphin County.....	10.00 to 16.58

While practically all of these have been appealed from, the Lackawanna and Luzerne County and Scranton City rates are absolutely confiscatory; in one case, where under the old valuation the taxes paid by the land owner equaled one-fourth of the average royalties received,

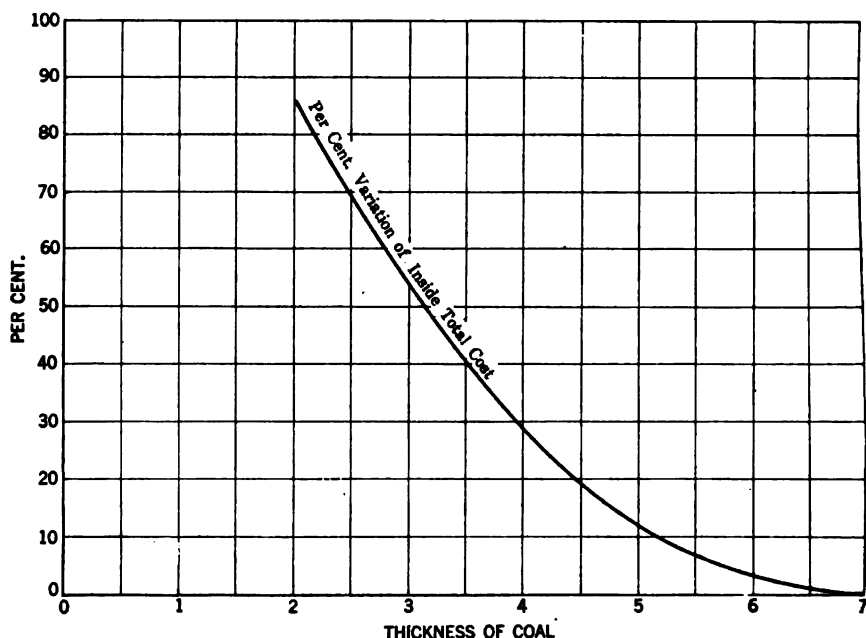


FIG. 1.—RELATIVE COST OF MINING COAL BEDS OF VARIOUS THICKNESSES.

the valuation was raised 563 per cent., making the lands worse than valueless to the owners.

Then, considering the various methods of coal-land taxation suggested, we find serious objections to all.

First: *Sale Valuation*.—Impracticable, from the impossibility of finding sufficient sales of typical lands for a basis of value, and further unfair in not considering the actual amount of coal in separate pieces of land, the sale value of a large tract of thick coal being entirely different from the selling price of a tract containing only thin beds, or beds partly exhausted, and the impossibility of separating the value of the coal from

the value of improvements and developments in the case of the sale price of operating collieries.

Second: Foot-Acre Valuation.—An illegal method, fairer than sale valuations, but presenting difficulties in determining a proper basis of valuation, and offering to unscrupulous politicians too easy a method for flat advances in valuation. This method fails to discriminate in the character of the coal assessed, not only in the thickness of the beds but in their constitution, whether large benches of clean coal or coal laminated with slate and bone in thin layers and of relatively little value. How much the thickness of beds may affect their value is shown by Fig. 1, the relative cost of mining various thicknesses of beds, compiled from averages of the actual costs in a large number of collieries in the Wyoming field.

Third: Royalty Valuation.—Illegal, and unfair from the fact that the rate of output and time of mining are overwhelming factors in the valuation. For example, assume five exactly similar properties, each containing 2,000,000 tons of coal, to be worked out seriatim at an average of 100,000 tons per year, and each paying 30c. per ton royalty, an annual royalty paid during the mining of each tract of \$30,000; on the basis of Luzerne County, Pennsylvania, 1913 assessment, these would each be valued at \$400,000, and would pay approximately \$8,000 annual taxes up to the average time of exhaustion. Their present values, on a royalty basis, calculated at 6 per cent. would be as follows:

Tract	Start Mining, Years	Complete Min- ing, Years	Present Value Royalties	Less Present Value Taxes	Net Present Value
First.....	0	20	\$344,100	\$58,880	\$285,220
Second.....	20	40	107,360	110,120	-2,760
Third.....	40	60	33,550	126,100	-92,550
Fourth.....	60	80	10,430	131,550	-120,650
Fifth.....	80	100	3,250	132,550	-129,300

Thus, even without any taxes, the coal held for future necessities is shown to have but small present value, and including the taxes to be paid, except the first assumed tract, to be mined out within 20 years, is worse than valueless, and would be held by the owners at a loss.

This condition may be well illustrated by Fig. 2, showing the present value, and present value of tax charge on \$100, to be paid at a future date, as on lands held for reserve.

Further, valuations on such a basis would result in absolute confiscation of lands leased on the low prevailing rates of the past, when the owners accepted royalties of from 8c. to 25c. per ton for prepared coal only, at present about 65 per cent. of the entire production, and further covenanted to pay all taxes assessed against the lands.

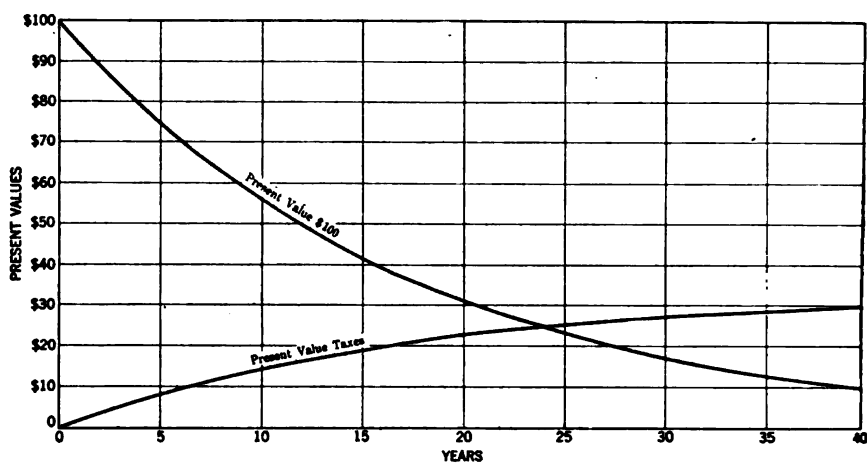


FIG. 2.—PRESENT VALUE OF \$100 FUTURE PAYMENT AT DIFFERENT PERIODS AND PRESENT VALUE OF TAXES ON SAME AT 20 MILLS FOR SAME PERIODS, AT 6 PER CENT.

Fourth: *Valuation on Estimated Profits*.—This method, which has been proposed but not used in the anthracite region, is manifestly illegal and most unfair because an accurate estimate of future profits is manifestly impossible; profits are dependent not only on the property, but

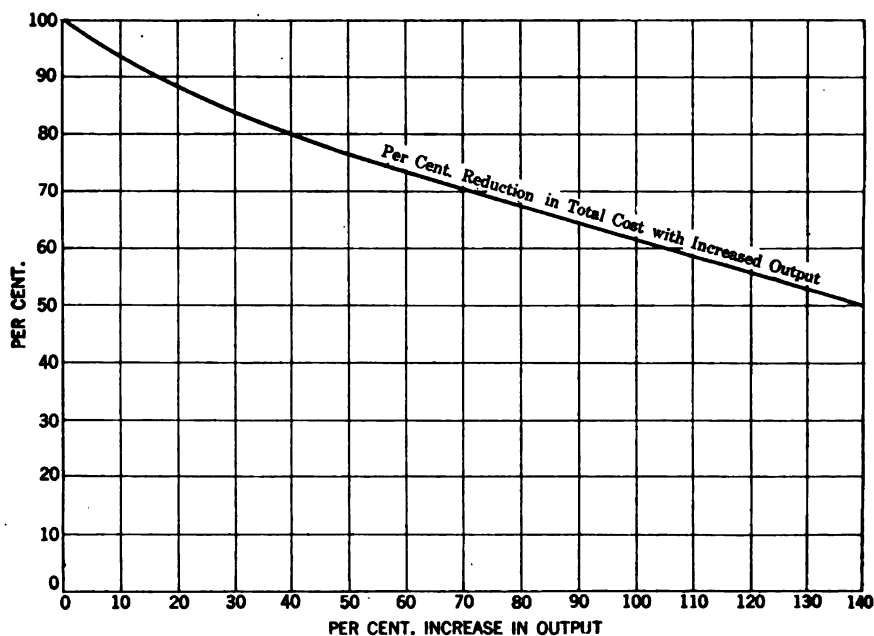


FIG. 3.—VARIATION IN COST OF MINING AT VARYING RATES OF OUTPUT.

on its management and on the rate of output, the latter at least partly determined by market conditions. That the rate of output is an important factor is shown by Fig. 3, showing the percentage variation in cost in a group of anthracite collieries under varying conditions of rate of output. Further, such a method of assessment is opposed to all ideas of conservation, as it penalizes good management and good mining by taxing brains as much as coal.

Conclusions.—It appears that none of the suggested or attempted methods of assessment has resulted, or can result, in an equitable valuation, fair and just to both the public and the owners of coal land; that even moderate taxation of the coal in the ground is opposed to all principles of conservation, as its effect is to put a tremendous premium on rapid mining, almost regardless of ultimate recovery, to encourage the destruction of poorer and thinner beds interstratified with the better ones, on account of the enormous penalty entailed in slower mining, and to discourage, by prohibitive penalties, the holding of lands in reserve for the future necessities of the people.

For these reasons it appears that the taxation of mineral in the ground is logically and economically wrong, leading to the rapid and uneconomical exhaustion of the mineral wealth of the country, and putting a premium on premature and wasteful exploitation, and that the proper method of taxation for all minerals would be a tax based on the value at the mine of each year's product at the local rate of taxation assessed for that particular year, including an assessment on surface lands, outside improvements, and machinery, the values of which are readily ascertainable, but not including any valuation of mine openings or inside improvements which are incidental to the mining process and which after the exhaustion of the mineral are of no value.

Thus, a colliery producing 1,000,000 tons of anthracite in 1912, with a value at the mine of say \$2,500,000, and with surface and improvements valued at \$1,500,000, should pay taxes for the year 1913 on \$4,000,000 valuation, regardless of the area of coal land tributary to such a colliery, and if for any cause the production in some later year should fall to \$1,000,000 in value, and the value of the surface and improvements decrease to \$1,000,000, the taxes for the next year should be assessed on but \$2,000,000 valuation.

This suggested method of assessment and taxation of coal lands is of course clearly illegal under the present laws of the State of Pennsylvania, and would require special legislation to put it in force, and while not absolutely just, in that it assesses coal from thin and impure beds, costly to mine, at the same rate as that from the more cheaply mined thick and pure beds, it would, if legalized, possess the inestimable advantage of doing away with all uncertainty and litigation as to assessed valuations; result in the payment of taxes in greatest amount at the times of greatest

production and consequent greatest population and public need for money; and by concentrating taxation on the land most actively worked, and relieving reserve land from its present crushing burdens, would tend to the conservation instead of the dissipation of the irreplaceable coal resources of the country, by encouraging the complete mining of lands once opened, including all workable beds, large and small, rather than the opening of the best beds on all lands to obtain immediate returns and avoid the burdens of accumulating taxation, even at the cost of the destruction of smaller and less valuable beds lying above the larger ones.

WILLIAM GRIFFITH, Scranton, Pa.—I have not read Dr. Chance's paper yet, but I desire to express myself as being entirely in accord with Mr. Norris's conclusions; also to add that not only is his method of taxation illegal, but there is no other sensible method of taxation of coal or any other mineral land in Pennsylvania that is legal. The only possible method you have now is to send the "ice man" on the property and ask him to guess a valuation on it. There is no possible method that any engineer, or anybody else who has any idea of the value of minerals, would use that would be sustained by the Supreme Court, because the law as it now stands does not recognize mineral value or any method of determining it beyond a mere guess. But the court and owners insist on appointing engineers to do this work and thereby warrant the inference that they desire it to be done by sensible men in an engineering manner. But after it is done, if the engineer in his report explains how he did it, it will be thrown out as being illegal under this antiquated law. That is the reason why we did not state in our Northumberland County report how we did it. We examined 100 or 200 different anthracite properties (there were two engineers and a layman on the Commission). In estimating the value, we used all the methods that Mr. Norris has referred to. We estimated the size, the quantity of coal in the ground, extent and condition of mine workings, its accessibility, its remoteness from market, the royalty, etc., and by combining all these things, and capitalizing the royalty for the length of time we expected the mine to last at a reasonable output, making proper adjustments in every case to meet varying conditions, we finally arrived at what we believed would be a reasonable value for each separate piece of land. But in our report we simply stated the value of each tract, giving the number of acres, without saying anything about how we got at it. And that is the only report that has gone to the Supreme Court and been accepted in the State of Pennsylvania. That report was objected to by the county authorities and our assessments were raised. It was then appealed to the Supreme Court, and after the Supreme Court stated that the foot-acre method was not legal, it added that there was no evidence in this report of the use of that method, and so confirmed that report. This was the only one in the whole State of Pennsylvania that has ever been confirmed by the

Supreme Court; so if any of you ever have occasion in Pennsylvania to value mineral property, do not tell the court how you do it.

S. A. TAYLOR, Pittsburgh, Pa.—An interesting matter came to my attention this week; while it is not exactly in line with this discussion, yet it is a matter that has some bearing on it. It is this: The Internal Revenue Act, providing the tax on incomes, recently passed by the national government, among other things changed the conditions as to the taxing of corporations, especially mines. Prior to this act becoming a law, corporations were permitted to deduct from their net income the sum of \$5,000, and an additional amount equal to that necessary to provide for the entire depreciation of their property.

The new law not only eliminates the flat deduction of \$5,000, but also provides that the deduction for depreciation of property cannot exceed 5 per cent. of the gross value or sales price of the product of the mines, at the mines. This does not permit the companies to deduct the whole of the depreciation of their properties in case that depreciation exceeds the 5 per cent. allowed by the law; for example, in this Pittsburgh district the average sales price of coal at the mines is approximately \$1.20 per net ton, 5 per cent. of which would be 6c. per ton, the amount permitted to be charged off as depreciation of property, while generally it would cost about 15c. per ton to replace that coal, and in many cases it would be 20c. per ton, which shows at once that this law is unfair to the coal operator of this district to the extent of from 9c. to 15c. per ton. In the Connells-ville coke district a very unfair condition is also shown. The royalty on coke alone is approximately 35c. per ton. The fair average sales price for coke this last year would be \$2 per ton; 5 per cent. of this equals 10c. per ton with which to replace the 35c., the actual value of the property.

The iron-ore mines of Michigan are more unfairly treated under the provisions of this new act than either the coal or coke companies above referred to. In many cases the royalty for iron ore will run as high as \$1 per ton. The sales price of the ore at the mine is about \$3 per ton, 5 per cent. of which is 15c., which will be allowed under this law for what costs \$1. I understand the iron-ore men have taken the matter up with the government for adjustment. The collectors of internal revenue acknowledge the unfair provisions of the law, but they are helpless to give any relief in the matter. It is thought by some lawyers that the only way this matter can be corrected is to have that part of the law repealed or amended, as the government cannot be sued and it is doubtful whether or not any decision of the Revenue Department can be legally made which will give equitable relief in this matter.

This whole affair shows what an injustice can be perpetrated upon the taxpayers by legislatures with little or no knowledge of conditions pertaining to the subject of their legislation.

R. D. HALL, New York, N. Y.—I think sometimes it is a little unfortunate that we hear about taxation problems almost entirely from the viewpoint of the anthracite region. The problem in that section of the coal-mining industry is largely different from that of the rest of the country. When Mr. Norris and others propose that the tax be put upon output, they do it, I believe, with a due sense of their responsibility to the community, and with the anticipation that under such a plan a fair amount of taxation would be paid by coal-land owners to the various treasuries; but, unfortunately, there are some people in other coal fields of this country who have large holdings and they would keep them for an indefinite time without use, if absolutely no taxation had to be paid until such time as they extracted the coal. As an example of such a large landholder who is desirous of holding land because he has to pay no taxes, I would bring to your attention "Uncle Sam," who puts a high price on all his holdings, and who is able to maintain that excessive valuation because under the law of the United States, Uncle Sam pays absolutely no tax on what coal lands he possesses. And I think not only he, but also several other smaller individuals, would be not a little disposed to hold on to their land indefinitely if you taxed their coal only on the output and they were able to hold it without any charge at all for taxation. It would be manifestly unjust to those people who are developing and working their coal and to those other citizens who are living in that particular district, whose property is more or less depreciated by the fact that the resources are not being used, if that man were allowed to hold his property without taxation until such time as he chose to develop it.

MR. GRIFFITH.—That may all be regulated by law, as in the case of unseated land at present. If a man owns a good farm and a portion of unseated land, he is not taxed on the latter the same as on the good land; assessors simply put a nominal value on the unimproved land, sufficient to let the owner know that the State appreciates that he has land there. To remind him that that part of his land may be of more value later they tax it every year, but not as much as they do his fine farm land. So with coal lands and mineral lands of any kind which are not being developed, there should be a nominal tax placed, sufficient to prevent holding it indefinitely for speculative purposes, but not enough to make it a burden to hold for future mining.

MR. EVANS.—Did I understand you to say that royalty could not be charged against operating expense?

MR. TAYLOR.—If you are paying royalty, you are permitted to charge off the royalty as the cost of mining. But if you unfortunately own the coal, you cannot charge it off. The government permits the royalty to be charged off as part of the cost of mining, and yet if you own the coal you will only be permitted to take off 5 per cent. of the sales price of the coal at the mine mouth.

Investigations of Coal-Dust Explosions

Discussion of the paper of GEORGE S. RICE, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2459 to 2492.

WILLIAM GRIFFITH, Scranton, Pa.—I notice in the discussions and literature that emphasis is placed on rock dust, shale dust, and limestone dust, and I infer that these different sorts of rock are ground up to produce this dust. Is that right?

MR. RICE.—That is correct, but we do not have to grind the limestone dust we are using in the experimental mine, as that is obtained from a quarry, and is used just as it comes; we can get it very cheaply, as it is a by-product of the quarries.

MR. GRIFFITH.—Have any experiments been made with ordinary dirt out of the fields; could not that be used as ground dust?

MR. RICE.—If you dry it and powder it enough so it will float in a strong air current it might be all right for the barriers, but if it is damp there will be a tendency for it to mat together; then it would be quite useless for making a dust cloud. There might also be another disadvantage if used through the whole mine, if it had very much free silica in it, which is bad for the lungs. Limestone and selected shale dust would not affect the health of miners.

MR. GRIFFITH.—In order to do very much good with any of those barriers, they ought to be rather extensively used throughout the mines, and there are few mines that have facilities for grinding up this rock into such a fine dust. If the ordinary earth from a plowed field could be used, even if it were not quite so good, it is more accessible and very much cheaper; therefore, in using it, with the universality of the supply throughout the country, it would tend to stop explosions.

MR. RICE (communication to the Secretary*).—In closing, it is well to again emphasize that I do not advocate barriers being a "cure-all" for explosions. It will be generally agreed that prevention is best, not only by means of improved methods of using explosives, and ventilation to prevent accumulations of firedamp, but also by either thoroughly wetting the dust throughout the mine, or else so treating it with rock dust or other inert dust that an explosion cannot start. From my observations I believe that rock dusting is a practicable method, and has the merit of visibility, and remaining effective for considerable periods, whereas the watering method has to be daily followed up, and if neglected for a short period the mine may become in a dangerous condition. The Germans

* Received Mar. 7, 1915.

attribute one of their great disasters to the fact that watering had not been done during the period of a Sunday and a holiday (that is, between Saturday and Tuesday), with consequent drying out of the dust. It is too delicate a matter to refer to specific disasters in this country, but it is thought that many similar instances have occurred in the United States, in mines ordinarily well taken care of, but in which, through temporary neglect of watering, the dust had become dangerous. Where the watering system is used it is undoubtedly a very great help to also sprinkle some rock dust through the entries, which will assist through its capillary attraction in wetting the dust, and will also help in making the dust pack together, so it will not be raised by a concussion.

Referring to Mr. Griffith's remarks, while earth from the fields could doubtless be used if dried, it may be pointed out that to take it from a plowed field would probably destroy ground valuable for agricultural purposes, and it would seem to be cheaper to take material of less value, such as limestone or shale.

Since the original paper was issued one of the largest coal companies is making a test of rock dusting in one of its mines, on stretches of the main haulage entry, and also in one of the butt entries. After first cleaning these roads limestone dust is being used. The conditions are very severe, as the cars are all leaky, and a great deal of coal is spilled off, particularly at the switches. After being down a month, at the time of writing this, according to the samples gathered and the analyses made, it shows most favorably, indicating that the limestone dust is still in sufficient excess to prevent a dust explosion from originating or propagating in these zones. It has the further advantage that the limestone dust is sticking to the ribs and roof so that it has the effect of whitewash, making it easy to illuminate the entry, which in itself is an element of safety. So far the test gives indications that the cost will be less than by the previous method of watering supplemented by calcium chloride; and judging from the rest of the mine it is believed that conditions are much safer where the rock dusting has been done.

Some interesting tests have been conducted at the experimental mine, in which the rock dust is not mixed with the coal dust but the coal dust is strewn on top. It makes a condition of maximum severity for the rock-dusting method. These tests, as yet limited in number, indicate that the rock dust is brought up into suspension and extinguishes the initial explosion nearly as well as if mixed before laying through the entries. The proportion of shale dust to Pittsburgh coal dust must be increased merely from 75 to 77 per cent. in the most extreme case, using wholly fine coal dust 80 per cent. of which passes through a 200-mesh sieve. With coarser coal dust mixed with the finer coal dust, which is the practical condition now under test, it is thought that less than 75 per cent. of shale dust will be needed to neutralize the coal dust spread on top of it.

Coal-Mine Explosions Caused by Gas or Dust

Discussion of the paper of HOWARD N. EAVENSON, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2637 to 2660.

GEORGE S. RICE, Pittsburgh, Pa.—I think it is very wise to point out the greater liability to explosions in certain parts of the year, if there be such difference, with the caution, however, that vigilance be not relaxed at other times. In comparing the figures for the United States with those of Europe, I think we should separate them as far as possible according to cause of origin and means of propagation. That is to say, it would appear to me that the majority of our explosions are due more directly to the ignition of coal dust directly from the flame of explosives, whereas, judging from my inquiries and observations in some of the mines of Europe, the majority of their explosions have been caused by the ignition of fire damp. Now, in the case of fire damp, I do not think that anybody has yet attempted or has been able to find any connection between the seasons and the issuance of methane into mines. One thing must be borne in mind, that if you get a strong explosion of fire damp, dust of a character that might not ignite readily by itself might assist in the extension of an explosion of fire damp through a considerable area in the mine, if not throughout the mine. While it is true we have many lesser explosions in this country attributable to fire damp, it would appear that the majority of all the mine explosions of the country, in considering the means of propagation beyond the locality of the origin, are due to coal dust; and it is the liability to coal-dust explosions that appears to be affected by seasonal changes. Hence the desirability of separating statistically (if it could be done by the inspectors) fire-damp explosions from coal-dust explosions, in order to obtain the true value of the seasonal effect. Nevertheless, in spite of the inclusion of fire-damp explosions with coal-dust explosions, the preponderance of the evidence, as indicated in Mr. Eavenson's figures for American mines, has shown that in this country there were more explosions in the winter and spring months, when the mines are drier; that is, when the mines are apt to be drier. Whereas, according to the European figures that he has gathered, it seems to be very clearly demonstrated that in France, England, and Belgium, the time when explosions occur has no connection with any particular season. That, I think, is partly due to the point just mentioned, that a majority of their explosions have originated from fire damp. The European mining men have been more careful, generally speaking, with the use of explosives than we have, so that the cause of ignition from that source has been less. Only comparatively few of the mines in foreign

countries permit the use of black powder. It is not permitted at all in France, Belgium, and Germany, except in a few of the shallow, non-dusty and non-gaseous mines. In Belgium, in mining coal in gaseous or dusty mines no explosive is used, and in rock work only permissible explosives may be used; but it must be admitted the non-use of explosives in coal is not a handicap in coal getting, since the mining is by the long-wall system and the roof weight brings it down easily.

Apart from the generally accepted fact that in explosions in European mines, fire damp, either wholly or in part, plays a most important rôle, an additional argument may be made as to why the seasonal changes appear to have so little effect in European coal mines, as seems to be indicated by Mr. Eavenson's figures, viz.: The European mines are so much deeper than American mines, and hence so much warmer, practically always warmer than the air at the surface, that there is little difference in the drying effect of the air current in either summer or winter. That is, the temperature of the air current by the time it has reached the working places has been so raised by the natural heat of the strata that it is always drying in effect even in summer time, and precautions are taken accordingly. On the other hand, in America the temperature of the mines is so much cooler, that in summer it is less than the average outside temperature, hence the entering air current, through being cooled, becomes fully saturated, and this in turn moistens the walls and ultimately the dust. Therefore there is for a few months lesser liability to have true coal-dust explosions.

C. M. YOUNG, Lawrence, Kan.—A few years ago the State Geological Survey of Kansas collected what was believed to be all the available information concerning coal-mine explosions, including causes, effects, and dates. This information was gathered from all parts of the world where records have been kept. After the statements of figures and facts had been tabulated, copies were sent for correction to the various countries from which the data had been obtained. When these corrected copies had been returned, it was considered that the list of explosions was as nearly correct as it could be made.

From the data in these lists it would be concluded, as Mr. Eavenson states, that there is no necessary connection between seasons and explosions.

Some possible reasons may be suggested for the increase in explosions mentioned in the paper. First, the change in the character of mine labor, with the partial disappearance of the miner who looked upon his labor with pride and had the attitude of the skilled artisan toward his work. Second, the increased use of explosives. It is true that the use of permissible explosives is increasing, and this fact decreases the danger, but there are still many districts in which black powder is used and its use is likely to continue for a long time. In many districts the ratio of

the amount of powder used to coal mined is increasing. The tendency to depend almost entirely upon explosives for breaking down the coal leads to the use of excessive amounts of powder. Third, the increasing tendency to shoot off the solid.

It is possible that one reason for the occurrence of many explosions outside the winter season may be found in the belief that they will occur almost exclusively in the winter. The belief that there is most danger of the occurrence of explosions in the cooler seasons may lead to carelessness in other seasons. In February and March the miner expects explosions, if there is any possibility of their happening, and at most other seasons he thinks that he is fairly safe, and it is possible that he is then a little less careful than he is in the winter months because of this belief.

H. H. STOEK, Urbana, Ill.—There is possibly another explanation or reason why we apparently have more explosions in winter than in summer, because our mines then work much more actively in many parts of the country. The European mines work more uniformly than ours, so that we might expect on that basis to have the explosions distributed throughout the year. Then there are some who think that when a mine is working irregularly it is more liable to explosions than when it is working regularly. Has Mr. Eavenson any figures showing connection between explosions and earthquakes?

MR. EAVENSON.—No, I have had no experience in the earthquake line, and I am mighty thankful to say I have had very little experience in the explosion line.

WILLIAM GRIFFITH, Scranton, Pa.—I think a question worthy of scientific investigation is whether the diffusion of gas in mines is in any way affected by the astronomical condition like tides in the ocean. In my experience, passing through the anthracite mines, I have for some time made it a practice to ask the inside men, fire bosses, mine foremen, and others, what time of day the most falls of roof occur, and I find that the great preponderance of roof falls in the mines occur at night time, *i.e.*, when the sun is on the other side of the earth. The question has arisen with me whether there is any possible effect from the attraction of the sun or moon, which would cause such a thing to occur or to effect the efflux of gas.

R. D. HALL, New York, N. Y.—People fail to realize, in comparing one country with another, that there must always be differences between them. England and Scotland lie against a great ocean, receiving all the time moist air with a percentage of saturation which often amounts to 100. That degree of moisture content is quite rare in the United States, and, moreover, the temperatures in Great Britain never get so low as those on this side of the water. Consequently, the air on entering the mines does not lose saturation so markedly there as it does here.

This makes the danger of coal dust less marked, and as it is the coal-dust explosion which is seasonal, there is not likely to be the seasonal difference in accident frequency in Great Britain which we have here.

Another cause for decreased danger from coal-dust explosions in Great Britain is the use of the long-wall system. When an explosion of any kind takes place in such workings there is plenty of room for the explosive violence to expend itself. It is rarely found in Great Britain that the violence is marked near the coal face. The explosion seems usually to start and work its principal havoc in the tightly gob-packed headings. Consequently the frequent use of the long wall probably decreases the frequency of coal-dust explosions.

And conversely long wall increases the probability of gas explosions, for the ventilation is frequently rendered inadequate where that system of mining is in operation.

But in regard to seasonal variations, we must not forget that the winter in Europe sets in earlier than here. In fact, it is quite usual in Great Britain to regard the last three months of the year as winter. Here the first three months are generally so classified.

W. H. GRADY,* Bluefield, W. Va.—One of the speakers made the point that perhaps the increase in explosions was due to the difference in supervision, due to the fact that they are employing Hungarians and Poles, rather than English and Welsh miners. If that is true, and I believe it is, the proper thing is to supply them as much supervision as that difference requires, and some mine operators have recognized the need of more and better supervision and are supplying it by safety foremen, patrolmen, or assistant foremen, or whatever they call them. The particular instance that has come under my observation is that of the United States Coal & Coke Co., at Gary, W. Va., where the death rate per 100,000 tons is one-fifth of what it is in the Pocohontas field. In speaking with different operators on that point, they sometimes say that increased supervision brings about increased cost; but with increased supervision, it is found that increased efficiency follows, and rather than an increase of cost, we have a reduction in cost and a reduction in the death rate.

R. V. NORRIS, Wilkes-Barre, Pa.—In connection with increased supervision, recently in the anthracite fields various large companies have been putting in a safety patrol system.

E. B. WILSON,* Scranton, Pa.—For the benefit of Mr. Stoek, we had an earthquake in Scranton some months ago, and I took occasion to write to all the operators to find if there had been an increase in gas, and

* Non-member.

they said no; that on the contrary, there had been less. I attribute that to a high state of the barometer.

F. Z. SCHELLENBERG,* Pittsburgh, Pa.—I believe explosions are apt to occur on Monday morning or after a holiday, when there is irregularity of supervision, or of the immediate examination of the working places. Some of the men who have had that responsibility are out, which is the cause to my mind of one of the largest catastrophies that we have had here.

S. A. TAYLOR, Pittsburgh, Pa.—A question has been in my mind for several months. It was stated to me by a man who had just returned from Europe, that a large explosion that took place in France last spring was caused by electric currents. On investigation, it was found that it was due to the electric waves caused by a wireless telegraph. Talking this matter over with one of the professors of mining in one of the schools not far distant from here, he told me that they had had something of the same experience, in connection with their wireless telegraph plants, and they were obliged to stop the operators. They ascertained that the two wireless stations set fire to a building which was half way between the two. Now, if anything can be evolved along that line, it seems to me we are treading on very dangerous ground when we allow the promiscuous installation of wireless telegraph systems. It seems to me it should be investigated by those who have the ability and power to do so.

HOWARD N. EAVENSON.—I think Mr. Rice's point, that if explosions could be separated into those caused by gas and those caused by dust, more dust explosions would be shown to have occurred in the winter months than in the summer months, is well taken. He knows, however, better than I do, the trouble one has in trying to separate these two classes of explosions from any records we now have. To illustrate this: I know personally of an explosion that was ascribed to dust, while it was a well-known fact in the vicinity that gas could be exploded by any miner in his working place any time after firing a shot.

There was no intention in writing this paper of making a comparison of the explosions due to seasonal variations in the different countries. My only idea was to present the facts.

As to Mr. Griffith's point about falls of roof: We prepared statistics for our 12 mines, covering the past 10 years, classifying accidents by the different days of the week, different hours of the day, and the causes—whether falls of coal or slate. The average number of accidents per day was practically the same throughout the week. There were a few more between 10 and 11 o'clock than in any other hour in the day. This does

* Non-member.

not exactly cover his point, as there might be many more falls at night than in the day time, when no accidents would result from them.

Regarding the explosion caused by wireless: The writer knows of a mine that had been shut down for about three years, which generated gas in small quantities. The fan and pumps were run two or three days once a month and were shut down the rest of the time. One afternoon during an electric storm there was an explosion in the mine, although there was nobody working in it at the time nor had there been for 18 days. The pumps were not running and there was no fire under the boilers. There was no electric current in the mine, as the trolley-wire connections were broken at the foot of the shaft. A careful investigation was made and the only cause to which this explosion could be ascribed was lightning, which probably ran down the shaft, possibly through the ropes, thence to the cage and to the tracks and followed the rails in to the place of the explosion. This place was an excavation being made for a stable when the mine was shut down, and which was driving up hill and was a dead end. Gas had evidently accumulated in here, and as the switches were all set to go directly to this place it was evident that the current had followed the rail into the gas and had then exploded it.

ROY PETERS.*—I was connected with a mine in Canada a few years ago. It was shut down on the 4th of July, and I took occasion to make some repairs on the shaft. Along about 3 o'clock in the afternoon a sharp flash of lightning struck the shaft. A few moments later black smoke came out of the mine mouth. We started up the fan and turned it off slowly to see what the effect would be, and it immediately cleared itself. After putting some air down in the mine we went down and investigated and found that this flash of lightning had gone down over the trolley wire and ignited the gas.

MR. NORRIS.—There have been two similar cases in the anthracite region of gas fired purely by the electric spark from a storm.

MR. RICE.—Regarding the difficulty that Mr. Eavenson mentions of getting accurate statistics, I think it is particularly true in the case of explosions. At the present time mine-accident statistics are collected by the various State inspection departments. The U. S. Bureau of Mines accepts these figures in its compilations for the whole country. The State inspection departments have no uniform system in that matter, and while some inspectors will call an explosion, the propagation of which is due to dust, a dust explosion, others will classify it on the basis of its origin. It is quite a usual thing in the Middle West to call an explosion a "windy shot," even if it has extended to the shaft. That is the cause of one of the difficulties we have in analyzing results. In statistics which

* Non-member.

are furnished to the Bureau of Mines, I have found many instances where what I would term a dust explosion has been called a gas explosion, windy shot, or powder explosion. I think all should try to obtain uniform methods in reporting; and, as regards explosions, I think we should classify them on the basis of their chief means of propagating, that is, gas or dust, giving the origins in a separate column under these classifications: Blown-out shot; overcharged shot; ignition of fire damp; electric arc or grounding; open flame; gob or mine fire.

Shot Firing in Coal Mines by Electric Circuit from the Surface

Discussion of the paper of GEORGE S. RICE and H. H. CLARK, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2563 to 2571.

NORMAN V. BRETH,* Pittsburgh, Pa.—It was my privilege to visit in the last two months on behalf of the U. S. Bureau of Mines 10 mines where electric shot-firing systems controlled from outside the mine are installed.

I will describe briefly what I saw in some of these mines, which are located in Utah, New Mexico, Oklahoma, Alabama, and Kansas.

At a mine in Utah, 230-volt alternating current is used to fire the shots, which is taken from a small transformer in the substation. The shot-firing circuit is independent of all other wires and is tested for grounds every day. There are no lightning arresters on the firing circuit. Porcelain insulators are used inside of the mine, and glass outside.

The switches are situated as follows: One in the substation, one in the shot-firer's cabin, two at each cross entry; where the shot-firing circuit crosses the trolley wire a switch is placed on either side of the entry in series with the shot-firing circuit.

The miners are checked in and out of the mine.

The coal is undercut and is shot down with permissible explosive, the shots being connected in multiple. There are 75 to 110 shots fired each day. A great many times there are no missed shots. If not more than two or three are missed they are left until next day. The fan is kept running at full speed when the shots are fired. The shot-firing switch is closed only once and the whole mine is shot at once.

At another mine in Utah, power wires are used for shot-firing circuits; 500-volt, direct current is used to fire the shots. The only sizes of wire used outside of the power wires are Nos. 8 and 14 B. & S. gauge. The return is connected to the pipe lines and track. Porcelain insulators are used inside of the mine.

The switches are situated as follows: One in power house, the shot-

* Non-member.

firing switch, one inside the mine controlling the whole mine, one at each cross entry, and one at the end of the trolley wire.

The shot-firing switch is only closed once when the shots are fired. Part of the shots are connected in series and part in multiple. The coal is undercut and shot down with permissible explosives. The miners are checked in and out of the mine. About 150 shots are fired each day. In case of missed shots a new hole is drilled. The whole mine is shot at once. The fan is left running when the shots are fired.

At a third mine in Utah, the power used for firing the shots is taken from a small transformer at 244 volts, alternating current. The main circuit is No. 6 B. & S. R.C., and the room wires are No. 14 B. & S. R.C. Porcelain insulators are used inside of the mine, and glass outside.

The switches are situated as follows: One firing switch, two at each cross entry, and a short-circuiting switch in by from the entry switches. The coal is undercut and is shot down with permissible explosives. The whole mine is shot at once; 200 to 300 shots are fired each day with about 2 per cent. of failures. The missed shots are fired the same night if possible. The shots are connected in multiple. The fan is stopped 5 or 10 min. before the shots are fired and started up immediately after. The men are checked in and out of the mine.

At a fourth mine in Utah, the power for firing the shots is taken from the switchboard at 500 volts, direct current. A pair of power cables is used for the main shot-firing circuit. Nos. 6, 10, and 14 B. & S. R.C. wire is used for the branch circuits. Porcelain insulators are used throughout the mine.

There are two switches at each point where the shot-firing circuit taps the power cables. There is also a shot-firing switch outside the mine. The shots are connected in multiple.

The coal is undercut and is shot down with permissible explosives, the whole mine being shot at once. The booster fan is stopped and the main fan is slowed down before the shots are fired. Missed shots are left until the next day. The men are checked in and out of the mine.

At a mine in New Mexico, the trolley wires are used for one side of the main shot-firing circuits, 250-volt direct current being used for firing the shots. The return is connected to the pipe lines and track. Nos. 8 and 14 B. & S. wire is used for branch circuits. Porcelain insulators are used in the mine.

The switches are situated as follows: One in the substation, one at the entrance to the mine, the shot-firing switch, one at the side track, one at each cross entry, and two at the end of the trolley wire.

The coal is undercut and is shot down with permissible explosives, the whole mine being shot at once. The shots are connected in multiple. The missed shots are left until the next day. The fan is left running when the shots are fired. The miners are checked in and out of the mine.

At a mine in Alabama the shot-firing circuits are independent of the other wires; 500-volt direct current is used to fire the shots. Nos. 4, 6, 10, and 14 B. & S. R.C. wire is used. Porcelain insulators are used inside the mine, and glass outside.

The main circuit is divided into sections by means of four double-pole switches. The first of these switches inby is provided with a pair of flexible leads 8 ft. long, as a protection against lightning.

The coal is undercut and is shot down with permissible explosives. The detonators are connected in multiple. The whole mine is shot at once. The fan is kept running when the shots are fired. The men are checked in and out of the mine.

At a mine in Kansas, a system, designed by the Scheitzel Shot Firing Machine Co., is used. This system employs a so-called "sparker," which automatically connects to the shot-firing circuit different sections of the mine, one after another. In some parts of the mine the shot-firing circuit is run under the ball of the rail.

The mine is shot in sections, 175-volt direct current being used to fire the shots. The coal is shot from the solid with black powder. The ignitors are connected in multiple. The main circuit is No. 10 B. & S. R.C. The cross-entry circuits are No. 12 R.C. duplex copper; No. 10 iron triple-braid weatherproof is used in the rooms. There are no lightning arresters on the firing circuits. The men are checked in and out of the mine:

At a mine in Oklahoma, the power used for firing the shots is 500-volt direct current. The main circuit is No. 6 B. & S. W.P. and all others are No. 12 B. & S. W.P. Wooden clamps are used instead of insulators. Lightning arresters are installed on the main circuit.

The switches are situated as follows: One in the shot-firer's cabin, one at the entrance to the mine, and one at each cross entry. The shot-firing switch is closed twice when the shots are fired.

The coal is shot from the solid with black powder. All shots are connected in multiple. The missed shots are withdrawn. The fan is shut down 20 min. before the shots are fired. The men are not checked in and out of the mine.

GEORGE S. RICE, Pittsburgh, Pa.—One point that I think is particularly commendable is the use of switches in the firing circuits, at the mouth of each room. It strikes me as a most excellent plan, because it permits the inspection of the switches by the foreman who might be traveling along the entry; that was one purpose for its inclusion in the suggested specifications. Another object was also sought, viz.: Where the miner connects up the shots, if he connected them up while the mine trolley and the other power lines were in use, then in case there was any accidental grounding he might have the shot discharged while still in the room, and possibly receive the direct effect from the blast.

I understand this has happened a number of times, but it would not be proper to state where; but by employment of an electric switch on the post at the mouth of the room, or working place, if this is kept open until the miner leaves his room, premature firing is prevented. Further, the position is such that it is plainly observable to any foreman passing along the entry. There is, of course, no necessity of such switches if the mine has no power lines entering it, and provided there are cutouts between the surface and the interior to prevent lighting from entering on the shot-firing wires. The plan of using room switches is also based on the idea that in most mines employing surface shot-firing the miner connects up the detonators with the shot-firing leads. I understand from Mr. O'Brien, General Manager of the Stag Cañon Fuel Co., Dawson, N. M., that the room switches have been taken from these mines; and I think it would be interesting if he would explain why this has been done.

T. H. O'BRIEN, Dawson, N. M.—Previous to this year the miners loaded their holes and connected the room shooting wires. Some years ago it was reported that on a couple of occasions the shots went off as the connection was being made. That the miners might be better protected, as Mr. Rice has stated, in case of grounding or accidental discharge of the shots while connecting the same, individual switches were installed at the neck of each room. These were closed by the miner on his way out. When he reached his room the following morning, his first duty was to throw out the switch. The foremen, fire bosses, and others made it a practice to see that this was done.

As a further precaution against accidents, the practice of the miners loading their holes was discontinued, and that duty is now performed by special shot loaders, who do not enter the mine until after the shift has been finished and all the men have been checked out.

As soon as the men are out, the power is shut off and the main switch at the mine entrance locked open. At no time is the current turned on while the shot loaders are underground.

The shot loaders take with them sufficient electric detonators for their respective districts. Under no circumstances are they to give or issue a detonator to any one. No pit boss or superintendent has authority to violate this rule.

After the shots have been loaded and connected in the rooms, the shot loader closes the entry switches on his way out. After all the loaders are out of the mine, the main switch is unlocked and closed. The shot loaders then go to the shooting cabin, which is always kept locked, and turn the current on the shooting line.

As soon as the shots have been fired the switches are thrown out and the shot loaders re-enter the mine for the purpose of inspection. When they reach the entries, the entry switches are thrown out and locked open.

They then proceed to the rooms. If a shot has missed the wires are disconnected and a note made of the missed shot. When the inspector returns to the surface the mis-shots are reported in a book kept for that purpose. The following morning, when the fire bosses are making their rounds, they also note any mis-shots and observe whether or not the wires have been disconnected. They also report in their book what they have found in this respect. In this way we have a double check on the mis-shots.

In the case of mis-shots the men are not allowed to go to work in that room on the following day. The reason for keeping them out is because of the possibility of their trying, by some means or other, to fire their shots, not only contrary to the rules, but in violation of the laws of New Mexico.

When any shots have missed, the electrician goes in on the following day and, with suitable instruments, ascertains where the trouble is. This is usually found to be a broken wire, possibly concealed by the insulation.

During the past two years I do not believe we have had more than two detonators which were defective. The shot loaders on the following night reconnect the holes.

It can be readily seen that under the system now in effect the accidental discharge of a shot while the diggers are in their rooms is next to impossible and that the room switches are not a necessity.

C. M. MEANS,* Pittsburgh, Pa.—It is apparent that an electrical system of shot firing from the surface is intended to take care of a condition that cannot be handled in any other practical manner. Such conditions are not met with in the Eastern States and the demand for such systems has been confined to the West and South.

The hazard to human life is lessened in such mines as employ the system, but the danger to property is not lessened and may be increased, because improper shots may be fired in this manner that a shot firer would not risk if fired from the inside of the mine.

A system as described usually represents considerable outlay for the first cost as well as maintenance. It must be properly installed and maintained or it may become a source of danger rather than a safeguard.

The voltage employed may be 250, but owing to the fact that no men are in the mine when power is on the line, there is no danger of accidental contact. The limiting feature is one of insulation and no difficulty should be experienced up to 650 volts. The higher voltage will admit of smaller conductors and should decrease the failures, as current of this higher voltage is more likely to go across dirty or loose connections.

Inasmuch as connections at the face will be made by the miners or shot

* Non-member.

firers, it is naturally inferred that many defective or poor connections will be made. The idea of using some sort of a mechanical connector has been considered but does not seem to be a practical solution.

The objection to a shot-firing system of this kind is the increased cost of coal, which may be anywhere between 5 mills and 3c. per ton. In many districts this increased cost represents a large percentage of the margin of profit. The possibility of failures is an item that is not figured, and may still further increase the cost.

The advantage gained is the increased safety to employees.

It is necessary to make a nearly complete analysis of a mine to determine definitely whether or not the introduction of a shot-firing system is the best method of reducing an existing hazard.

MR. RICE (communication to the Secretary*).—The last remark by Mr. Means touches a most important matter. It was not the thought of the authors of the original paper that the system of electrical shot firing under discussion was the ideal system in all cases; in fact, they frankly acknowledge it is in the nature of a last resort. The idea of firing from the outside of the mine implies there is danger in allowing any one to be in the mine.

In mines which are subject to instantaneous outbursts of gas, either inflammable, or carbon dioxide as in central France, there is no choice, in the interest of safety, to firing the shots from the surface. It will also be universally conceded that it is the safest plan in shaft sinking, or in approaching an underground body of water. Happily, great instantaneous outbursts of gas are practically unknown in this country, so the reason for any extended application of the system arises from danger of coal-dust explosion through blasting coal off the solid with black powder or dynamite, frequently with excessive charges. When the coal can be and is undercut or otherwise freed on the bottom, top, or middle, and permissible explosives within the charge limit are used, there is practically no danger from shots originating explosions and therefore no occasion for firing from the surface so that there need not be any one in the mine. Why the former practice is adhered to is too long a story to take up here, but the fact remains that in a majority of the coal mines in the central West and Southwest, shooting off the solid with excessive charges of long-flame explosives is employed, with a heavy loss of life among the professional shot firers who do the shooting when all the other employees are out of the mine. The record of one Southwestern State shows that the average length of service of a shot firer until he is killed by an explosion originating from a shot that he, or one of his companion shot firers, has ignited is but $2\frac{1}{2}$ years.

There is one feature mentioned in Mr. Breth's description of the meth-

* Received Mar. 7, 1915.

ods used in the mines he visited when shot firing from the surface is employed, that attracts attention, though not strictly related to the subject in hand: namely, the closing down or in some cases slowing down of the fan during shot firing. When a mine is small, it is possibly a good precaution to cut out the fan momentarily to lessen the strain of concussion on the fan, but there does not seem to be any need in large mines such as Mr. Breth examined. On the other hand, if a mine makes any fire-damp there is serious danger from the practice. The frequency of explosions in the Southwestern districts, out of all proportion to the output of coal of those districts, would seem to fully refute the claim of lessened danger of explosions by shutting off the ventilating current at shot-firing time.

GEORGE S. RICE and H. H. CLARK (communication to the Secretary.*)—In the original paper, which presented some tentative ideas regarding certain details of shot firing by electric circuit, the practice of connecting shots in series in any one working place was advocated, primarily on the ground that if one shot depended on the successful blast of the adjacent shot to lighten the burden of the former, or what is known as a "dependent shot," it was important that either all should go or none. Otherwise if a dependent shot was discharged and the relieving shot was not, the burden of the former would be too great and this would probably result in a blown-out shot, with possibility of starting a dust explosion, which would damage the mine. This recommendation was also based upon the fact that if detonators are identical in construction and electrical resistance they can be shot through a considerably greater line resistance when they are connected in series than when they are connected in parallel.

However, experiments made since the paper was written have shown such a tendency on the part of detonators to shoot at different time intervals that the practice of connecting shots in series would probably result in missed shots if the voltage were scarcely enough to cause detonation; or if, as is probable, the application of voltage to the detonators is gradual and not instantaneous, as might be supposed.

At the instant that the firing switch is closed the voltage at those points in the mine that are most distant from the source of power is less, and possibly considerably less, than the voltage that is supplied to the entry nearest to the source of power. The voltage equalizes throughout the mine as the shots are discharged in room after room and entry after entry, but rapid as this process is relatively, it may be sufficiently gradual to cause the more sensitive detonators to fire and the less sensitive detonators to miss if the shots are connected in series. Hence it is believed that to accomplish the purpose sought of minimizing the risk of explosions, the arrangement in parallel is best.

* Received Mar. 10, 1915.

Gasoline Locomotives in Relation to the Health of Miners

Discussion of the paper of O. P. HOOD, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2607 to 2611.

R. V. NORRIS, Wilkes-Barre, Pa.—I have had a little experience with gasoline locomotives and I want to indorse very strongly Mr. Hood's point with regard to the danger of carbon monoxide poisoning. I think the terrible danger is not fully realized.

I was foolish enough some time ago, in investigating a gasoline locomotive, to attempt to walk out of the mine entry behind it, and for several days I felt very seriously the carbon monoxide effects.

The use of gasoline locomotives ought to be confined to mine entries where there is a very large amount of air. One of the grave dangers is their great convenience and the temptation to run them into chambers for just a few moments to get a car or two.

GEORGE S. RICE, Pittsburgh, Pa.—Apparently one of the great dangers is in the style of the gasoline engine. When my attention was first directed to the type of gasoline locomotives which I saw in the foreign mines—France, Belgium, and Germany, I was most favorably impressed with them, and they have given no trouble on account of fumes, so far as I could learn. Large numbers of them were being used, but all were of small capacity, and of a kind in which the speed control is by a governor of the hit-or-miss type, the engine running continuously. Their power capacity was very low, if I remember correctly it was not over 4 or 5 h.p., but they were used under favorable conditions where the grades in the mines were slight, since the entries or tunnels had a slight grade in favor of the load, making about the same pull for the empties against the grade as for the load with the grade. I understand the hit-or-miss valve arrangement enables a setting that will give more perfect combustion than can be insured with the valve controlled by the operating man, as in the automobile type of valve and engine used in this country. Apparently the European type of locomotive does not have the capacity to permit use under the conditions existing in this country, where most of our entries or tunnels have variable grades and long heavy trips of mine cars are used.

S. A. TAYLOR, Pittsburgh, Pa.—I have had a little experience along this line in a mine of low vein coal where just such conditions as Mr. Rice speaks of were prevalent. We had great difficulty with the motor and did not use it except on return-air courses, using our main heading as the return-air way. We had no good method of regulating the amount of gasoline admitted into the engine. {This seems to be one of the weakest points in all of the gasoline locomotives I have examined. When the

motor stalls or checks up a little the man running the motor will throw in excessive gasoline, and the fumes are extremely hard to handle even with a very good current of air. In order to overcome the stalling we took up the track and relaid it to a uniform grade, but even then, when there was an extra wagon—a ton or two on a trip—we would find the same difficulty arising. We operated our fans and pumps with electricity. On one occasion the generator broke down, and while we were waiting for a piece to replace the broken part the sump began to fill up; as the pump was in there the motor was run in quickly to pull it out, with disastrous results. Two men were working at the pump. No air current was passing, and when the motorman started the motor a lot of gasoline was turned on, with the result that one man was overcome almost instantly, requiring several minutes' work to relieve him, and our mine foreman, who was not nearly so susceptible to the fumes, felt the effects very perceptibly. One of the main things, I believe, is to get a device on the motor that will control in a very much better way the amount of gasoline admitted into the engine.

Another thing of importance is to employ capable men to run it. In these days of automobiles, you would think that would not be a very serious question, but it is around a mine. You can put on one of your best machinists to run the motor; if it stalls, he gets rattled about his machine not working, and if he has no automatic control he will throw on too much gasoline, and trouble results.

W. G. WHILDIN, Lansford, Pa.—We had an experience of more than a year with five gasoline locomotives, 10-ton type, of three different makes. We found that it was not so much a question of the health of the miners, because the men were not affected at all, but the health of the company's finances, which at that particular plant suffered very much. We finally put in an electric haulage system which we had been religiously keeping away from on account of its greater cost.

After the installation of the electric haulage system, about four months ago, we found our output increased 30 per cent. from exactly the same openings.

The main trouble with the gasoline locomotives was that they were not dependable. In the morning all five of them would start out on their runs in fine condition, and sometimes—in fact, very often—within a few hours, three and sometimes four of them would be out of service, and trips stalled in all parts of the mine. As a result, the locomotives which were still working were overloaded by having to move these cars and their regular trip in order to clear the road.

As I stated, our men were not bothered at all by the fumes from the locomotives. Our workings are all on heavy pitches, and by actual tests men working in blind ends were not affected even when the motors were located directly below them. We had one case, however, where four

motormen were so badly overcome that the use of the pulmotor had to be resorted to. This occurred at a point near the foot of the shaft where we had the locomotives stored over night, and when the engines were started in the morning the fumes were very strong.

So far as care of the locomotives was concerned; I want to say that at all times they received the best care possible. Every night two high-class repairmen, assisted by some or all of the motormen as was found necessary, overhauled the locomotives.

O. P. Hood, Pittsburgh, Pa.—Since the discussion has been confined to the difficulties that have been met with, I feel like defending the gasoline locomotive. The great difficulty has been that people do not recognize the presence, the source, or the magnitude of the danger in using these machines.

It would be a bad thing to place powder in the hands of those who do not know its danger, but simple rules and regulations allow its extensive use in proper hands. The same can be said of gasoline locomotives. That there is danger from the exhaust gases is not recognized until after some unfortunate experience such as has been related in the discussion. The limits are not known—just how far one can go and still be safe—and for this reason investigations were made by the Bureau of Mines. It is believed that under proper regulations and with intelligence the machines can be safely used. The gasoline locomotive can be easily overloaded and evidently with disastrous results, as related, but some regulation resisting this tendency should not be hard to enforce when it is made evident that it is necessary for safety.

The makers of the gasoline motor are of the opinion that a really good carburetter such as you would expect to have on your automobile is not suited for mining machinery. Gasoline locomotives use but one make of carburetter, its great advantage being ruggedness and simplicity. Five or six per cent. of carbon monoxide may be made with these carburetters, while in certain long-distance economy runs of automobiles with better carburetters the exhaust gases contained only a fraction of 1 per cent. This shows what it is possible to do with gasoline locomotives.

The gasoline engine in a mine should be frankly recognized as a possible source of danger and the necessary conditions for safety provided. When the danger is thus recognized the conditions for safety are not difficult of attainment in many places.

A New Safety Detonating Fuse

Discussion of the paper of HARRISON SOUDER, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2547 to 2556.

CLARENCE HALL,* Pittsburgh, Pa. (communication to the Secretary†).—The introduction of Cordeau detonant to replace electric detonators used in blasting operations in this country has not met with the same success as it has in Europe. This has been due to the fact that the explosives manufactured and sold here are far more sensitive to detonation than the explosives used abroad. In Europe, and particularly in France, the explosives used in blasting operations are of the ammonia nitrate class and can only be detonated by the use of Cordeau detonant or a powerful detonator. Practically all explosives used in mining operations in this country are of the nitroglycerin class and when they are in good condition, and properly used, incomplete detonation rarely occurs.

In quarry operations, where the large drill hole method is employed and the charge of the explosive sometimes extends a distance of 30 ft., it has been found that one electric detonator will not always produce complete detonation of the entire charge in the drill hole; but when three or more electric detonators are employed throughout the charge, misfires or incomplete detonation rarely occurs. Cordeau detonant would no doubt be applicable in work of this kind, but in other blasting operations, where small charges of explosives are used and when sufficient stemming is employed to confine the charge, it is not believed that it could be advantageously or economically used. The cost of this detonating fuse is about 10c. to 15c. per yard, and it is necessary to use an amount that will extend from the bottom of the drill hole up to and beyond the mouth. It is true that this additional cost is a small part of the total cost of blasting in quarry operations, and the fact that this fuse will always insure complete detonation warrants its use in work of this kind.

The rate of detonation of Cordeau detonant is about 4,900 in. per second, but it cannot be assumed that greater energy will be developed when used with dynamites having high rates of detonation. However, when detonating fuse is used with dynamites of low rates of detonation, a greater shattering effect may be produced on the surrounding rock. When dynamites are used having rates of detonation higher than this fuse, then the only advantage in its use will be to insure complete detonation. In regard to increased efficiency resulting from the use of Cordeau detonant, this may obtain when an explosive of low rate of detonation is used or when explosives are used which have aged and are not in good con-

* Non-member.

† Received Nov. 19, 1914.

dition. The statement that 20 per cent. reduction in powder charge could be made when the detonating fuse is used is questionable. The tests that the writer has made with Cordeau detonant do not show that greater efficiency obtains when used with small charges of explosives, such as the permissible explosives used in coal mining, or with other explosives in operations where shallow holes are employed.

S. P. HOWELL,* Pittsburgh, Pa. (communication to the Secretary†).—In discussing Mr. Souder's excellent paper on Cordeau-Bickford, a new safety detonating fuse, I shall have much favorable comment to make, for I believe that because of the safety features of this kind of fuse there is a field for its use in quarrying and in open-pit mining operations.

Among the advantages resulting from the use of this fuse, I desire to discuss particularly the following:

1. The elimination of the detonator or the electric detonator from the charged hole.

2. The reduction of "blind" misfires.

3. The increase in the disruptive effect of low-grade high explosives.

4. The possibility of using relatively insensitive explosives.

The advantages mentioned are discussed in order below:

1. Of the three materials, trinitrotoluene (the explosive in the detonating fuse), 40 per cent. "straight" nitroglycerin dynamite, and mercury fulminate composition (commonly used in detonators and electric detonators), the least sensitive to impact and friction is trinitrotoluene; while the most sensitive is mercury fulminate composition. Bureau of Mines tests with the pendulum friction machine¹ show that a sample of trinitrotoluene is not exploded with a carborundum shoe operating over a steel anvil. The sensitiveness of the mercury fulminate composition is well illustrated by the many instances of accidents caused by pricking with a pin or needle the composition in a detonator. Consequently, the substitution of the detonating fuse for the detonator or electric detonator in drill holes is clearly a move toward safety, as the latter devices, because of their sensitiveness, have been held responsible for the many drill-hole accidents which would not have occurred had the holes contained nothing more sensitive than the ordinary high explosive.

2. In open-pit mining or excavating, and in quarrying, though misfires seldom occur, the possibility of their happening is always imminent. The "blind" misfire—one that is not known to have occurred—is clearly the kind most likely to cause an accident, the workmen being ignorant of the presence of the unexploded explosive, and, perhaps, of the detonator or

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† Received Dec. 5, 1914. Published by permission of the Director of the U. S. Bureau of Mines.

¹ See Hall, Clarence, and Howell, Spencer P.: Tests of Permissible Explosives. *Bulletin No. 66, U. S. Bureau of Mines*, Plate I, and pp. 15 and 16 (1913).

electric detonator. The presence of an effective detonating agent—the detonating fuse—throughout the entire length of a charge in a drill hole must reduce, or perhaps eliminate, these “blind” misfires.

3. The disruptive effect² of high explosives depends in a measure upon the rate of detonation of the explosive. Tests at the Bureau of Mines show that the rate of detonation of a low-grade high explosive is increased to the rate of detonation of the detonating fuse extending through it. The following table of rates of detonation of certain samples of high explosives and of detonating fuse shows which particular high explosives may have their disruptive effect increased by the use of the detonating fuse.

Rates of Detonation of Trinitrotoluene Detonating Fuse and of Certain Samples of High Explosives

Class and Grade of Explosive	Diameter of Explosive Tested		Rate of Detonation	Authority
	Inches	Milli-meters	Feet per Second	
Trinitrotoluene detonating fuse.....	¹ / ₁₆	4	13,750	Bureau of Mines.
Trinitrotoluene detonating fuse.....	¹ / ₁₆	6	15,170	Bureau of Mines.
Trinitrotoluene detonating fuse.....			17,000	Manufacturer.
60 per cent. "straight" nitroglycerin dynamite.....	¹ / ₁₆	32	20,490	Bureau of Mines.
50 per cent. "straight" nitroglycerin dynamite.....	¹ / ₁₆	32	18,850	Bureau of Mines.
60 per cent. strength low-freezing dynamite.....	¹ / ₁₆	32	16,640	Bureau of Mines.
40 per cent. strength gelatin dynamite.....	¹ / ₁₆	32	16,210	Bureau of Mines.
40 per cent. "straight" nitroglycerin dynamite.....	¹ / ₁₆	32	15,650	Bureau of Mines.
30 per cent. "straight" nitroglycerin dynamite.....	¹ / ₁₆	32	14,920	Bureau of Mines.
40 per cent. strength ammonia dynamite.....	¹ / ₁₆	32	10,350	Bureau of Mines.
5 per cent. granulated nitroglycerin powder.....	¹ / ₁₆	38	3,340	Bureau of Mines.

* The data for Bureau of Mines rates of detonation for the explosives are taken from *Bulletin No. 48, Bureau of Mines*, p. 44.

4. As the percentage of nitroglycerin in low-grade dynamites is decreased, their sensitiveness to detonation is decreased, and hence when the nitroglycerin content has been reduced to a certain point their use becomes commercially impracticable. However, it would appear that such explosives, though having a low nitroglycerin content, could be used if the trinitrotoluene detonating fuse were used to detonate them. This would be likewise true of very insensitive explosives which contained no nitroglycerin.

Now, because this detonating fuse, as used in mining operations, is new in the United States, it seems highly desirable to emphasize such of the characteristics of the fuse as are most likely to cause failure when the fuse is improperly used, for this material, in common with other new materials used in mining operations, is likely to be misused.

² See Hall, Clarence, and Howell, Spencer P.: *The Selection of Explosives Used in Engineering and Mining Operations. Bulletin No. 48, U. S. Bureau of Mines*, pp. 41 to 44 (1913).

Mr. Souder has emphasized (see p. 2555) the importance of not joining the main fuse to the branch fuse; of joining the branch to the main at right angles; and of using the crotch method of attaching the branch to the main. Investigation has shown that the detonation of a branch will not detonate a main with certainty, and this confirms results obtained in Bureau of Mines tests^{*} in which attempts were made to detonate the fuse through its lead covering with No. 6 and No. 8 electric detonators, but without success.

The Bureau of Mines has tried various methods of attaching the branch to the main and has not been able to improve on the crotch method, while the simple method of attaching the branch alongside and parallel with the main resulted in failure.

By reason of the lead covering that incloses the trinitrotoluene of the fuse the material is rather heavy. For instance, a sample of 6-mm. fuse weighed 45 g. per foot, or approximately 10 lb. per 100 ft., and a sample of 4.3-mm. fuse weighed 27 g. per foot, or approximately 6 lb. per 100 ft. A sample of the same 6-mm. fuse 12 in. in length did not support a weight of 34 lb., and a sample of the same 4.3-mm. fuse 12 in. in length did not support a weight of 21 lb. It, therefore, appears that when the fuse is lowered into the deeper holes (50 ft. or more) care must be exercised to prevent jerking it, lest its own weight cause it to separate.

For use in these deeper holes the manufacturers have produced a covered fuse. The cover consists of closely wound cotton thread and adds greatly to the tensile strength of the fuse, so that lengths much longer than are feasible with the uncovered fuse can be used without danger of breaking. Except for the covering, the two types are identical.

Care should be exercised in tamping a hole containing the fuse and in choosing the stemming, for roughness in tamping may break the fuse, especially if the stemming be of hard and angular material.

HARRISON SOUDER (communication to the Secretary*).—The Institute is fortunate in persuading Messrs. Hall and Howell to discuss the use of detonating fuse, since their work with the U. S. Bureau of Mines gives an authoritative character to their opinion in regard to explosives.

Notwithstanding Mr. Hall's statement to the contrary, as a matter of fact in average daily American practice, misfires and incomplete detonations are quite common in both large and small drill-hole blasting and are the cause of many accidents.

A misfire, especially in the case of large well drill holes, is a very expensive proposition and adds a large element of danger to quarry work, where the steam shovel is used in loading the broken rock.

* Received Mar. 7, 1915.

^{*} See Hall, Clarence, and Howell, Spencer P.; *Investigations of Detonators and Electric Detonators. Bulletin No. 59, U. S. Bureau of Mines, pp. 22 to 24, (1913).*

We know from our investigations that our own blasting troubles at Cornwall were typical of blasting troubles in many large and well-managed limestone and cement-rock quarries, with this exception: that our troubles were increased by the greater possibilities of the electric current being short circuited by the magnetic iron ore.

It is particularly for the large or well drill-hole blasting operations that we recommend Cordeau detonant. For our small drill-hole blasting we still prefer the electric detonator. While the use of Cordeau may not increase the energy of dynamite of equal or higher explosive rate, yet as a matter of fact it does increase the energy and efficiency of the total mass of dynamite in the hole by bringing about an instantaneous explosion of the whole charge, the effect being the same as if we had placed a continuous line or string of detonating caps from one end of the charge to the other.

The increased efficiency and decreased cost of blasting is made possible in that, with Cordeau detonant, a lower-strength and less sensitive explosive can be used to do the same work as an equal amount of higher grade, exploded in the old way; or the amount of high-grade explosive necessary under former methods can be reduced by using Cordeau in place of electric detonators, amounting in our Cornwall practice to about 10 per cent. saving. The saving of 20 per cent. is claimed by M. Fongarelles of the Lens Mining Co., operating French coal mines, and does not obtain here.

Furthermore, the time required to load a large blast is nearly cut in two by using Cordeau, and this is an item of great importance as it reduces expense as well as the hazard.

Mr. Howell in his discussion of the paper brings out the main features that should be emphasized, viz.: the first and most important gain from the use of Cordeau is safety, and incidentally there is at the same time a saving in expense.

It not only eliminates the electric detonator from the charge and places it outside the drill hole, where it is under observation and can be removed in case of trouble, but it does away with electric lead wires around the mine or quarry, where live current is used in blasting. In some recent instances special power equipment was installed to provide current for this blasting. This expense is eliminated.

The use of old blasting batteries, which are generally overrated and overloaded and are a fruitful cause of misfires, is done away with. Blind misfires are almost impossible in open bench work. Under the old method the presence of wires from the detonator might mean that the detonating cap had exploded properly or it might mean that the detonator was still buried in a stick of dynamite under the rock, waiting to be struck by a pick or dipper tooth, whereas the presence of a piece of Cordeau indicates at once a misfire.

The rates of detonation given by Mr. Howell indicate that the rate for Cordeau is about equal to that of 30 and 40 per cent. nitroglycerin dynamite, which is the grade of dynamite in common use in quarries. The French experimenters claim a higher rate, as do also the manufacturers.

As to the methods of using Cordeau, we are free to confess that we have had some troubles in using it. The greatest trouble came from the inferior quality of the Cordeau which was imported from abroad. The trinitrotoluene compound, or T. N. T. as it has been abbreviated, was not stable and it was found after a while that it turned from pure white to yellow brown. This condition could be disclosed by a fresh cross cutting or by peeling some of the lead casing from the inclosed explosive.

The manufacture of this material is now carried on in this country since which time trouble from this cause has ceased. Nevertheless, each lot is inspected before being used.

Another trouble was due to the breaking of the Cordeau in loading especially in deep holes. The American makers immediately made it with a cotton sizing or covering over the lead, greatly reducing liability to break. Occasionally, some defective lead covering is discovered. This renders the fuse liable to break easily and also permits moisture to enter.

The original Cordeau used was 6 mm. in diameter and an effort to reduce the cost was made by reducing the size to 5 mm. This was found unsatisfactory and about 5.5 mm. is now generally recommended.

Our feeling is that for big shots 6 mm. diameter is best, especially if the crotch method of uniting branch and trunk lines is used.

Some trouble was encountered in making the crotch connection of a branch line with main leader. The extreme care necessary for this operation, especially with the smaller diameter Cordeau, was not always exercised. Also it consumed much time. After many experiments on a small scale and actual trial in large blasts, the method given below was evolved and has proved on the whole most satisfactory for ordinary practice.

After the holes are loaded, allowing about 10 in. of Cordeau to project from each hole (in case of a single row), a brass ferrule is placed on the end of each fuse, and a No. 7 blasting cap is inserted in it to bear against the end of the fuse. Then a half stick of $1\frac{3}{4}$ in. diameter 40 per cent dynamite is slipped over the end of each branch fuse, so as to inclose the No. 7 cap. Then the main lead Cordeau is strung along the holes and notched into each half stick of 40 per cent. dynamite at the end of each branch fuse; and the connection is mud capped or covered over with a little mound of clay or sand and patted down. This is carefully done for each hole. Then the two ends of the lead fuse are brought together and a No. 7 detonating cap attached to each end, and these are inserted in a stick of 40 per cent. dynamite. Then a piece of ordinary cotton fuse, say

2 ft. long, is tipped with a detonator and this inserted in the stick of dynamite, which is then mud capped. When ready the fuse is lighted in the ordinary way and sets off the whole charge. In case two rows of holes are shot at one time, as is our ordinary practice, the connections are made so as to insure simultaneous explosion of all holes as nearly as possible, since

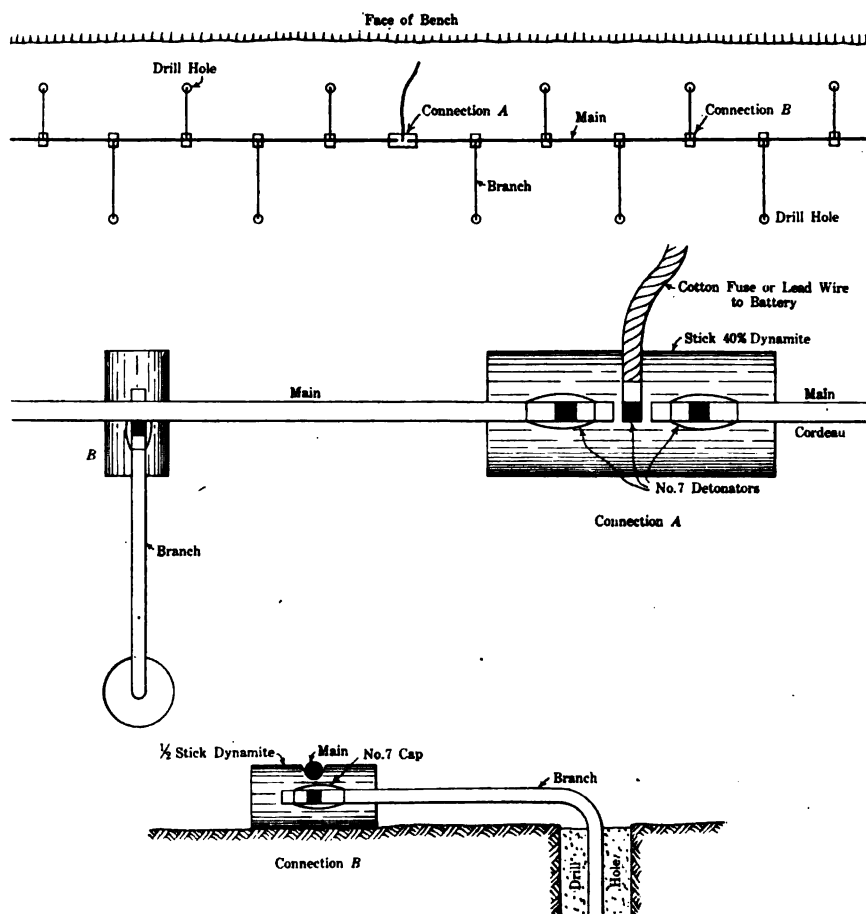


FIG. 9.—PRESENT METHOD OF USING CORDEAU.

we found that the short interval of time between the explosion of the back row and the front row, in cases where the back row went first, permitted rock strata to move sufficiently to cut off the front row of fuses in some cases. Where many rows are blasted at once many cross connections should be made between the main trunk leads.

Where a limited number of holes have been shot we have modified the practice, attaching to the Cordeau from each hole an electric exploder,

wiring the holes in series, and shooting with a battery in the old-fashioned way.

The sketches, Fig. 9, show the method we now use.

In conclusion, we may state that in our experience and judgment the introduction and use of Cordeau detonant marks a great advance over former practice, in making for safety and economy in blasting operations, especially for well-drill or churn-drill deep-hole blasting.

The use of Cordeau detonant has now become quite customary in the large cement-rock quarries of eastern Pennsylvania, and we believe further investigations of this material by the U. S. Bureau of Mines would be of much service and value to those operating open-cut mines and quarries.

A Test of Centrifugal Motor-Driven Pumps

Discussion of a paper of S. S. RUMSEY and W. F. SCHWEDES, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 94, October, 1914, pp. 2613 to 2635.

K. A. PAULY, Schenectady, N. Y.—I have seen this installation and I wish all those interested in the question of electricity as a mine pumping agent could see it. It is not only a model for mine installation, but might well serve as a model for any electrically driven pumping plant. I think I am correct in saying that the pumps are the largest electrically operated centrifugal pumps in mine service in America. I talked with Mr. Rumsey in Duluth just before coming here, and he told me that the actual tests of the equipment as installed checked very closely with the shop tests; that the water delivered by the pumps was slightly in excess of the guarantee, something like 3 or 4 per cent., and that no troubles except of a very minor nature had been experienced with the installation from the start.

One point which has been brought out by Mr. Rumsey, and by others who have looked into this question, is the marked saving which can be made by the application of electricity to pumps, especially to the main sump pumps. The pumps are always located a long distance from the boiler plant and the losses in the steam mains are large. While many of the steam pumps are of a very high grade, yet the tests seemed to indicate that the steam consumption is extremely large. The reduction in operating cost of one small installation which I have in mind pumping against a head of approximately 2,000 ft.—which by the way was a reciprocating pump—reduced the pumping cost to the neighborhood of one-fifth of the previous steam cost. The high steam cost was probably due to leakage in the engine and the loss in transmission in the mine, the pumps being located almost half a mile from the boiler plant.

From the standpoint of the manufacturers, I can say we will be very

glad to hear from all of the operators as to the possibilities in electric pumping, and the conditions which should be given particular attention, especially with reference to high-head installations.

WILLIAM KELLY, Vulcan, Mich.—I do not know how many here are interested in the subject of mine pumping. To those who are, it is an exceedingly important matter. The water in the mines has to have the first consideration, for if the pumps do not run nothing else is done. Where the quantity of water is considerable they are going night and day, Sundays and holidays. The selection of electrically driven pumps involves not merely a comparison between electrical pumps and steam-driven pumps, but there is a question also as to the type of electrically driven pumps—whether they should be reciprocating or centrifugal pumps. From the experience that has come to me it would seem as if each has its appropriate field. If the quantity of water is small, the reciprocating pump is doubtless the proper one to use, but when the quantity increases materially there is more of a choice and the efficiencies of the two types of pumps become more nearly equal. Efficiency as shown by a test is not always the controlling point, because tests are usually made when the machinery is working at its full capacity, but in regular pumping the supply of water is usually more or less irregular. Under such conditions there has to be some arrangement to allow a part of the water to run back, or the pump has to be stopped and started, with an adequate place for holding the water when the pump is idle. Then too comes up for consideration the difference in first cost and the interest charge, so that in each installation of mine pumping if electricity is to be used the conditions must be carefully considered and will determine the type of pump to be selected.

GEORGE S. RICE, Pittsburgh, Pa.—It has been a little difficult for me to get my bearings in the matter of the relative merits of the reciprocating pump and the centrifugal. I made my first visit to the mining districts of Europe in 1908, and I was very much astonished to find how universally the multistage electric-driven centrifugal pumps were being used at that time when they were so rarely used in mines in this country. I think in no mines in Germany or France did I find the reciprocating type. In view of the conservatism generally in England and in some districts visited it surprised me to find that that was the situation, and I had wondered why in this country our mining people, who are so generally advanced in the introduction of mechanical devices, had been—I won't say behind, but, let us say, lagging, in view of the recent developments in the introduction of the multistage pump. The latter seems to offer such very great advantages. It is interesting to note regarding the Delaware, Lackawanna & Western mine water shaft in the vicinity of Scranton, which has automatic arrangements for lowering and reversing,

the water tubs being in balance, and which I think was hoisting about 10,000 gal. a minute about 450 ft., that in subsequent enlargement of the drainage plant they put in centrifugal pumps of larger capacity, and I believe ultimately expect to supplant the hoisting plant.

In regard to the point that Mr. Kelly has brought out, that he thinks that where the quantity of water is small the reciprocating pump still holds its own: I wonder why this situation cannot be met in mining practice by running the centrifugal pump intermittently, and holding the water in sumps in the intervals, the advantage of this arrangement being conditioned on the greater efficiency of the multistage pump, at least over the old type of small mine pump.

O. P. HOOD, Pittsburgh, Pa.—One of the interesting things about deep mine pumping is the inertia effect of the column of water in the discharge column. On account of the heavy pulsations, the pipe line has to be built so much stronger than is required by the static head of the water that it has added largely to the cost. Now I am asking whether in these centrifugal pumps, where the heavy pulsations are absent, the operating cost of pumping is cheapened or not.

MR. KELLY.—I cannot answer that question exactly, but it is very evident that with centrifugal pumps there is an absence of vibration in the discharge line, which is very noticeable of course in some other pumps.

MR. PAULY.—I can perhaps give a little information in this connection. I know that this matter has been and is being thoroughly investigated by one or more of the large valve manufacturers, one of whom told me a short time ago that he had a valve which looks very promising for eliminating a large part of the water hammer which takes place with high-head reciprocating pumps. However, any device of this kind which is subject to failure is not to be recommended.

With reference to operating pumps intermittently, this is largely a question of sump capacity and is quite common practice. I know of one company in Michigan which operates its large pumps but 8 hr. per day.

I think the electrical manufacturers are largely responsible for the slowness with which electrically driven centrifugal pumps have been adopted in this country. However, it is to a certain extent up to the mining men. In Europe the mines are worked to a point where the returns are smaller per ton and every advantage must be taken of making savings, while in this country the mines are newer and in many cases the ores are richer; the original equipment installed when the mine was opened up is still intact, and the mining men themselves have not become, until recently, so intensely interested in economy as is the case in Europe. As to the manufacturer of electrical equipments, he has been doing a mail-order business, so to speak, in other lines of application. The

sale of electrical equipment to the large industrial plants has been accomplished with little effort and has afforded such a wide market for purely standard apparatus that the electrical manufacturers have not until recently gone after the mining business. Further, cheap and reliable electric power has not until very recently been available in most of the camps, and the high cost of isolated plants has limited their installation.

H. H. STOEK, Urbana, Ill.—Possibly an explanation of why the first centrifugal pumps were discredited more or less is the fact that the technical press was filled by oversanguine manufacturers with prophecies that the new form would entirely replace the reciprocating. Then a few accidents and failures did a great deal to take away the credit from the new pump.

A most interesting hydro-electric plant is located near Clausthal, Germany, inside of a mine. There the water is not looked upon as a drawback, but the head is being used to drive electrical machinery. I wonder whether there are places in this country where such an arrangement would be advantageous.

MR. KELLY.—When centrifugal pumps with high heads were first tried in this country they had not been perfected in detail. An installation at Butte was a failure on account of serious difficulties with the thrust bearings, and thus the whole system came under condemnation. Afterward that difficulty was entirely eliminated, and centrifugal pumps greatly improved in other particulars, but some prejudice still remains. Then, too, there is this, that in making tests at full capacity, the reciprocating pump will show a higher efficiency, but I very much question whether in a very large number of the installations they are getting in practice the efficiency which the test at full capacity shows. In practice the centrifugal pump, if in good order, will very often equal the general working efficiency of the other type of pump.

W. A. THOMAS,* Pittsburgh, Pa.—There are two points that have not been brought out particularly clearly, and one is the fact pertaining to Mr. Rice's question—the critical point as to head on the centrifugal type of pump. In the average coal-mining operation a pump is required which is adaptable to a wide variety of places in a mine where the heads may vary a great deal; care must be used in operating a centrifugal pump under a low head for the reason that under a half head it will tend to overload the motor seriously, and slightly over the head for which it is designed it will deliver no water at all.

Another point: in this country the reciprocating pump manufacturers have possibly made a departure which has not been made abroad, in

* Non-member.

getting away from steam designs for electric drive. The steam-driven pump would have a crank speed of 40 to 50. Some of the manufacturers of electric pumps have decreased the size of the plungers and increased the size of the valves to get a permissible water speed through the valves and increased the revolutions of the pump to from 65 to 75, which has greatly facilitated the electric drive, because of the great reduction that had to be made from the steam speed to the motor speed. The matter of efficiency is still open for discussion and will be as long as the two types are built, which will be perpetually. Undoubtedly the quintuplex pump has the highest efficiency. The quintuplex pump manufacturers will guarantee as much as 87 per cent. efficiency. The greater number of plungers in the reciprocating type of pumps tends to reduce materially the water hammer, and that, combined with the compression chamber, has to a great extent reduced the added cost of the plunger-pump installation. Against these factors, however, is the simplicity and low cost of the turbine type of pump, which makes it popular where definite conditions of head and water are known, but it is difficult to get fractional loads, and it resolves down to the question of making a sump arrangement for gathering the water.

MR. KELLY.—The turbine pump is a pump which is particularly suitable for a fixed head, but can be varied as to quantity. The reciprocating pump, on the other hand, can be varied greatly as to the head, but should have its full quantity of water—is not that true?

MR. THOMAS.—That is true to a certain extent; but the reciprocating pump lends itself to by-passing to get fractional loads. With the centrifugal, a slight speed difference will tend to make a great quantity difference.

R. V. NORRIS, Wilkes-Barre, Pa.—In the anthracite region, we have used probably every type of pump, and the centrifugal pump, we have found, is particularly applicable to conditions where the quantity of water is practically constant. With lower than the rated amount of water the efficiency of the centrifugal pump drops off rapidly, and it is necessary to run at intervals, which means supplying outfit for the maximum operations, and greatly adding to the cost. We still use a great many steam pumps in the anthracite regions.

MR. RICE.—I would like to ask Mr. Norris a question as to the expense when the water is acid. Under those conditions is the centrifugal pump more economical to use than the reciprocating?

MR. NORRIS.—I do not think it is. For very acid water I know of no instance where a centrifugal pump is used.

EDWIN LUDLOW, Lansford, Pa.—In the anthracite mines of the Lehigh Coal & Navigation Co., located in the Panther Valley, in the

southeastern corner of the anthracite field, we have extremely hard water conditions, as our large veins, running from 50 to 100 ft. in thickness, standing on from 45° to a vertical pitch, cause heavy surface breaks all along their outcrops on both sides of the valley, and intercept all the flood waters during heavy rains.

Our average pumping amounts to 25 tons of water to a ton of coal hoisted, and our vertical lift is from 250 to 1,000 ft.

At nearly all of our mines, we have installed water hoists, which are steam driven with 42 by 64 double-cylinder engines; they are able to hoist during the wet season a 3,600-gal. tank a minute from our 1,000-ft. levels.

At one mine, we have a drainage tunnel $4\frac{1}{2}$ miles long, starting from the Lehigh River and running to our Nesquehoning shaft, intercepting it at the third level.

During a severe flood, we gauged the amount of water coming out of this drainage tunnel and found that it was flowing at the rate of 40,000 gal. a minute. This mine is located where the flood conditions are the worst. With the opening up of our mines and the letting of the water down to the deeper levels, the problem is becoming one of increasing cost each year.

The acid contained in the water has made it necessary to line the pumps and column pipes with wood, and for that reason the water hoist has been found the cheapest in annual upkeep.

We have equipped several pumps recently with cement-lined working barrels and cement-lined columns and find that "Manheim cement" withstands the acid, and has proved entirely satisfactory during the year and a half that these pumps have been in service.

We have also used a cement-lining on slush pumps pumping the slush from the breakers to settling dams, and the cement has held up in that case.

On account of the heavy first cost of installation for either steam-driven or electrically driven plunger pumps, owing to the large amount of space which they require under ground, we are now experimenting with centrifugal pumps, made of all bronze and as acid resisting as possible, to be used in flood times, to assist the water hoists when the rush of water is too great for them to handle.

The largest of these pumps was built for 3,000 gal. a minute on a 750-ft. head and is 10-stage. It was not expected that these pumps would be able to maintain their efficiency in handling acid water if run continuously, but as their service will be intermittent and probably not continuous over a period of more than two months in a year, it was felt advisable to experiment with them, as the space they would require would reduce the first cost of installation to about one-half that required with plunger pumps of the same capacity.

MR. PAULY.—I would like to ask a question with reference to the operation of centrifugal pumps operating under variable heads at constant speed. We have recently quoted on a constant-speed motor to drive a centrifugal pump, and I believe the pump has been purchased, the head on which ranges from zero to approximately 800 ft. I have been endeavoring to get a line on the pump efficiency, operated under these conditions, but have not been able to do so and would appreciate some approximate figures at varying heads.

MR. KELLY.—It is possible to design a centrifugal pump so that the impellers can be changed. Impellers of a different diameter can be put in, which will serve to vary the capacity of the pump as to the head.

MR. PAULY.—This is a sinking pump to which I refer and I do not think that the impellers can be changed, although I am not certain of this. The pump has been purchased in Germany, I believe, and we know little concerning it except as to the limiting dimensions to which our motor must conform.

S. S. RUMSEY and W. F. SCHWEDES, Duluth, Minn. (communication to the Secretary*).—As the two 6-stage pumps have been in operation since July, 1, 1914, their performance in actual service may be of interest.

The pumps are installed on the 12th level, 965 ft. below the surface, as is also the Venturi meter measuring the discharge. The wattmeter measuring the input is located in the substation on surface, and it records not only the power used for pumping, but also that lost in the cable between the substation and motors, and that consumed in lighting the pump station.

In the first three columns of the accompanying table are shown the actual readings of the Venturi meter, the wattmeter, and pressure gauge in the discharge end of the pump, from which is figured the overall efficiencies in column 4. In column 5 is shown the estimated power consumed in the cable loss and pump-station lighting, which is not chargeable to the pump performance. Column 6 shows the actual overall efficiency of pump and motor as revised in accordance with the estimated losses.

Month	Actual Vent. Meter Readings Total Gal.	Total Input Kw-hr.	Average Dynamic Head	Overall Effic.	Kw-hr. in Cable and Lighting. Not Chargeable to Pumps	Net Effic., Per Cent.	Time in Operation Hr. M.	Time in Operation, Per Cent.	Average Gal. per Min.
July.....	118,326,000	523,800	991	70.25	3,480	70.70	685 5	92.1	2,876
August.....	105,470,000	460,805	991	71.15	3,400	71.68	648 30	87.2	2,708
September.....	98,000,000	424,105	991	71.86	3,260	72.40	605 20	81.4	2,696
October.....	122,090,000	543,040	991	69.91	3,490	70.34	675 35	90.8	3,014
	443,886,000	1,951,750	991	70.72	13,630	71.22	2,614 28	87.8	2,830

*Received Nov. 16, 1914.

Inasmuch as the 16th-level pumps had not been placed in operation we were unable to get a continuous supply of water for the 12th-level pumps; therefore, they have operated intermittently during the four months, and on account of the insufficiency of water have been operated at about 58 cycles instead of 60.

Comparing the quantity of water indicated by the Venturi meter with the quantity estimated from the plunger displacement of the reciprocating pumps as recorded by revolution counters, with liberal deductions made for slip, we feel positive that the Venturi meter is registering between 2 and 5 per cent. less water than is actually pumped. As soon as possible we will calibrate the Venturi meter, which has not been practical up to the present time.

Tin and Coal Deposits of the Fu Chuan District, China

Discussion of the paper of M. B. YUNG, presented at the Pittsburgh meeting, October, 1914, and printed in *Bulletin* No. 93, September, 1914, pp. 2451 to 2458.

T. T. READ, New York, N. Y.—I might point out that this paper of Mr. Yung's is on a field that is comparatively little known. It is not as well known as the Kochiu tin field, of Yunnan, which is probably the third in tin output of the world. This field under discussion is considerably further west and the tin which is produced here is, according to Mr. Yung, of somewhat higher grade than the Yunnan tin, and is used by the refiners at Hong Kong to improve the grade of the Yunnan tin. Most of you know that the tin from the Yunnan is shipped to Hong Kong, where it is refined and shipped to Europe. Mr. Yung points out that the Fu Chuan deposits are entirely alluvial and not found in any veins. They differ in that from the Kochiu deposits, which Leclere says are "nullement alluvionels; leur origine filonienne est de plus evidentes." Granite is the country rock here; doubtless the tin occurred in veins in the granite, and the deposits have resulted from the disintegration of the granite. This is a country of strong erosive agencies. The country is semi-tropical, and the rainfall is heavy, so that one would naturally expect deep surface erosion of this kind.

It is interesting to note in connection with the coal field mentioned that this is only one of many coal areas in China; one of the most fortunate countries in the world, in the wide distribution of its coal. Coal occurs almost everywhere. It has this disadvantage, however, that it is usually very friable. The strata have generally been subject to disturbance, so that as compared with the coals of this country a great deal of the coal is slack, while our coal contains much lump. The natives have met this difficulty by the general use of the practice of coking. Aside from

the big anthracite field in Shansi, the coal throughout the country is mostly bituminous, varying from toward lignite to semi-anthracite. There is very little lignite, except in Manchuria. The Chinese have coked the coal for centuries so that it will stand transportation, since the coal is generally transported in baskets. Fine coal would all filter through, and if they started with a ton, they would arrive with possibly half a ton, so that coke-making is very generally practiced by the natives, in crude furnaces. Now, with the organization of large companies, they have built modern coke ovens. At Pinghsiang there are two large banks of Otto ovens, and the Chinese Engineering & Mining Co. at Tongshan is about to let a contract for building ovens. There are also coke ovens at Penhsihu, in Manchuria. China has tremendous resources in coal which are certainly bound vitally to affect the welfare and the future of the country.

GEORGE S. RICE, Pittsburgh, Pa.—I would like to ask if the anthracite deposits in China have been sufficiently exposed to know whether it is a uniform body of anthracite like that in eastern Pennsylvania, or isolated patches such as in Colorado and New Mexico and Washington.

MR. READ.—Drake has estimated the average workable thickness as 22 ft.; the seams are almost horizontal and occur at no great depth over an area which Von Richthofen estimated at 13,500 square miles.

MR. RICE.—What is the age of the deposits?

MR. READ.—Usually they are found in the upper part of the Carboniferous. The Chinese coals average a little higher in the geologic series than our coals in this country. There is a great deal of difficulty in the correlation of strata. But my impression is that they are all rather higher in the scale than our own. They are probably in the upper Carboniferous or Permian. Some of the anthracite coal is the best I ever saw. Some of it is very hard and other varieties are very free burning; and it is fairly cheap. The mining is very cheap; the transportation is what makes it expensive. That is being remedied by the tremendous mileage of railroads which is now being built there. E. W. Parker, in one of his papers, estimated the tonnage of anthracite available for mining as approximately equal to the anthracite resources of the United States.

MR. RICE.—Is there any volcanic recognized in it?

MR. READ.—I think not.

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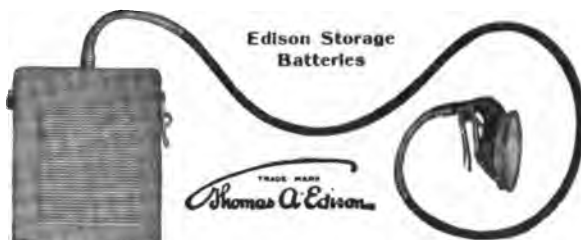
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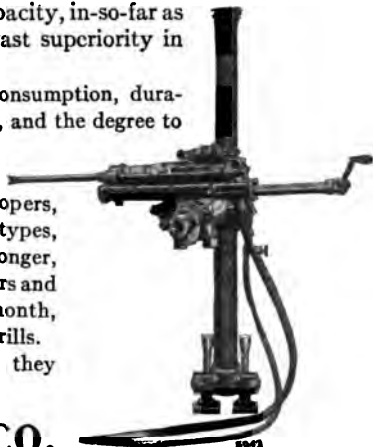
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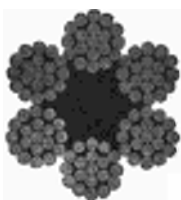
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